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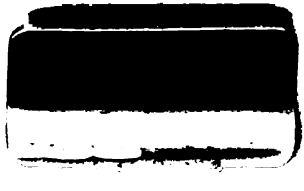
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THE

METALLURGY OF LEAD

AND THE

DESILVERIZATION OF BASE BULLION.

BY

H. O. HOFMAN,

PROFESSOR OF METALLURGY, MASSACHUSETTS INSTITUTE OF TECHNOLOGY.

SEVENTH IMPRESSION.

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PREFACE

TO THE FIRST EDITION.

In 1870 Dr. Percy published his great work, "The Metallurgy of Lead, Including Desilverization and Cupellation," which has become the standard book for England and America, and is also highly esteemed by the Germans and the French, into whose languages it has been translated. It contains an exhaustive discussion of smelting and desilverizing as carried on in the principal European works. During the past 20 years, however, so much that is new has been introduced in American practice that a book embodying the latest improvements seems almost a necessity for the student. It is the aim of the present work to supply this need. Thus, while European practice is not at all excluded from the book, the main object has been to make it a guide for Americans, and European methods have been discussed more in connection with their applicability in this country than from any other point of view. In the subject of the blast furnace, for instance, only such features have been brought out as seemed useful for America. Reverberatory furnace practice, on the other hand, which has not made much progress as yet in this country, is given more in the European way, while the ore-hearth treatment follows both English and American methods, which supplement one another. Three classes of readers have been kept in mind—the student, for whose use the ground principles and many definitions are given a prominent place; the metallurgist, who needs minute detail for his practical operations; and the investigator, who will find in the footnotes the principal references for the subject in its various branches.

To insure the latest data a number of important German works were visited in 1890 and the representative American ones in 1891, and the information obtained has been embodied with the name of the authority given, whenever this has been permitted. The author desires to thank all who have assisted him in this way and otherwise to amplify his notes, gathered through many years of practical life.

H. O. H.

Massachusetts Institute of Technology, May, 1892.

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PREFACE

TO THE SECOND EDITION.

As the first edition of this book only appeared in December, it is impossible to do more for the second, which has been called for in an unexpectedly short time, than to add a list of *errata*. The author will be grateful to readers who will call his attention to any still uncorrected errors, whether of type or otherwise, or who will favor him with suggestions through which his work can be improved in each new edition. A treatise on any technical branch of science must necessarily represent to a great extent the personal experience and conclusions of the writer. Thus he is very liable to emphasize difficulties or advantages familiar to himself in a given mode of practice at the expense of others with which he has not happened to be brought in contact. It is only by comparing the experiences and views of others with his own that any one can hope to produce in the end a book of general and lasting value.

H. O. H.

Massachusetts Institute of Technology, January, 1893.

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PREFACE

TO THE FIFTH EDITION.

SINCE the first appearance of this book six years ago there have been so many important changes in the smelting of silver-lead ores and the refining of base bullion that a new and enlarged edition has now become a necessity. In order not to increase too much the bulk of the volume, all discussion of the dry and wet assay of ores and metallurgical products has been omitted. This could be done the more readily as several treatises have appeared within a short time with full details of the quick methods in common use.

The most important change in the book is the new chapter on the roasting of ores. Perhaps it may be said that more attention has been given to mechanical roasters than is appropriate to a treatise on the metallurgy of lead. But lead smelters to-day treat all kinds of silver and gold-bearing ores, which are often rich in copper, lead being only the collector of the precious metals, and thus encroach quite unavoidably upon the field of copper smelting. Therefore the subject of mechanical roasting seemed to demand considerable notice. At the present time a reaction has apparently set in against it; the writer cannot but believe, however, that this is only temporary. The chapter on the properties of lead and of some of its compounds contains all the new facts published since 1892. In the short review on ore deposits the many references to sources of information have been omitted, because several treatises giving full details have recently appeared. A large number, however, of technical analyses of ore shipments from the different districts have been added and the references given. The chapter on sampling and purchasing of ores will be found to be much altered. The F. L. Bartlett process for treating zinc-

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lead sulphides has been described, as it has many points of interest for the lead smelter, although it does not, strictly speaking, belong to a treatise on argentiferous lead smelting. The chapter on smelting works has been greatly enlarged, some of the leading plants of the country being described more in detail than is usual. The chapter on the smelting of ores in the blast furnace has been made to include all the changes of detail which have been so numerous within the past few years. In desilverization, steam pattinsonizing has received special attention on account of the increasing prevalence of bismuth in the ores now treated and the consequent possibility that it may in the future be combined with the Parkes process in order to produce a larger percentage of corroding lead than can now be obtained. In the Parkes process the improvements by W. H. Howard have greatly changed the practical working. In cupelling, the typical English furnace has undergone radical changes in construction. In the annual volumes of "The Mineral Industry" the writer gives a summary of everything that appears in technical literature relating to silver-lead smelting. Some of the material collected for this purpose has been used in the present volume, but most of the new matter was obtained by personal visits to the leading works and the discussion of details with superintendents and managers. A tour for this purpose was undertaken in the summer of 1896, and the writer desires to acknowledge with many thanks the uniform kindness and courtesy bestowed upon him at that time. He feels especially grateful to Messrs. L. S. Austin, F. L. Bartlett, A. S. Dwight, C. Eilers, E. F. Eurich, W. H. Howard, M. W. Iles, C. H. Livingstone, E. G. Mathewson, F. B. F. Rhodes, R. D. Rhodes, F. D. Weeks and H. A. Vezin, with whom he came into personal contact, as well as to many others with whom he corresponded to obtain the necessary information from places which he was unable to visit.

H. O. H.

Massachusetts Institute of Technology, Boston, Mass., September, 1896.

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PART I.

INTRODUCTORY.

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CHAPTER I.

HISTORICAL* AND STATISTICAL† NOTICE.

§ 1. Introductory Remarks.—Lead was probably known at a very early date. The oldest people of whom we have any record—the Egyptians—used it in glazing their pottery, and the abundance of silver among the ancients suggests the presence of argentiferous lead ores in many places. We know that lead was mined in considerable quantities by the Greeks and Romans. The mines of Laurium, in Attica, reopened by a French company in 1863, flourished especially in the fifth century B.C. The Romans extracted large amounts of ore from the mines in the southeastern part of Spain, which were opened in the third century B.C. by Hannibal, and form the main lead district of Spain to-day. They also carried on mining operations in England and along the Rhine from Bâle to Cologne. About the year 1000 A.D. the celebrated German silver-lead mines of Saxony, Silesia, and the Harz Mountains, and those of Austria were discovered.

§ 2. Lead in the United States.—At the present day the lead mines of the United States occupy the second place, Spain being the leading producer. In this country lead mining dates as far back as the beginning of the seventeenth century, when lead was mined and smelted near Falling Creek, Va. During colonial times lead mines were operated mainly in North Carolina, New

* Pulsifer, "Notes for a History of Lead," New York, 1888.

† "United States Geological Survey: Mineral Resources of the United States, 1882," *et seq.*, Washington, 1883; "The Mineral Industry: Its Statistics Technology and Trade," 1892, *et seq.*, New York.

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York, and the New England States, but on a small scale, and not very successfully. The mines principally mentioned are the Washington mine, Davidson County, N. C.; the Rossie mine, St. Lawrence County, N. Y.; and the mines near Middletown, Conn., and Southampton, Mass. None of them were worked continuously, and at present there is no lead ore produced in the East except a little in North Carolina and Tennessee.

The lead ores of Missouri were discovered by Le Sueur in 1700 or 1701, and were first worked in 1720; of these the Mine La Motte, of Madison County, which is worked to-day, was the first discovered. Rumors of lead in the upper Mississipi Valley were afloat as early as 1766, and in 1788 Dubuque obtained from the Indians a grant for a lead mine on the place where the city named after him now stands.

From these two districts the bulk of the lead of the United States came until the first great mines of the West were opened in 1867. The total product of the United States in 1825 was only 1,500 tons. This increased steadily until 1848, when 28,000 tons were produced. After falling off considerably for a little over 20 years, the production of the Mississippi Valley increased again, and reached 50.468 tons in 1898. As the total product of the United States for the same year was 228,475 tons, this district produced 22% of the entire output. All the rest came from the Western States and Territories, which produce argentiferous lead ores, while those of the Mississippi Valley are practically free from silver.

Argentiferous lead was first produced, according to Hahn, in 1866 or 1867, near Helena, Mont., and at Oreana, Nev. In 1869 the mines of Eureka, Nev., discovered in 1864, were opened, and the treatment of ores was begun in the following year. Next came Utah, where smelters were erected in 1870, followed by Colorado, which came into prominence in 1878. Later Idaho, New Mexico, and Arizona were added to the list. Colorado has been since the largest producer, and it together with Idaho now (1900) produces about 50% of all the lead of the United States.

§ 3. THE WORLD'S PRODUCTION.—The world's production in 1900* was as follows:

* "The Mineral Industry," vol. ix.

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LEAD PRODUCTION OF THE WORLD. (IN METRIC TONS.)

Year.	Austria.	Belgium.	Canada.	France.	Germany.	Greece.
1894	7,570	14,120	2,587	8,758	100,753	14,534
1895	8,085	15,573	7,467	7,627	111,058	19,845
1896	9,769	17,223	10,977	8,223	113,732	15,180
1897	9,860	17,023	17,698	9,916	118,881	16,486
1898	10,340	19,330	14,477	10,920	123,742	19,196
1899	9,736	19,500	10,982	11,200	129,225	19,058
1900 (a)	9,500	19,500	12,000	11,000	130,000	19,000

Year.	Hungary.	Italy.	Japan.	Mexico.	New South Wales.	Russia.
1894	2,113	19,605	1,454	57,000	50,000	743
1895	2,277	20,353	1,950	68,000	38,000	412
1896	1,911	20,786	1,958	63,000	30,000	262
1897	2,527	22,407	1,737	71,637	22,000	450
1898	2,305	24,543	1,705	71,442	50,000	560
1899	2,166	20,543	1,700	84,656	70,000	500
1900 (a)	2,200	21,000	1,700	97,612	80,000	600

Year.	Spain.	Sweden.	United Kingdom.		United States.	Totals.
			Foreign Ores.	Native Ores.		
1894	152,620	330	9,538	30,162	145,906	617,780
1895	160,786	1,256	16,036	29,464	142,298	650,580
1896	167,016	1,530	25,889	31,311	158,479	676,602
1897	189,316	1,480	13,312	26,988	179,369	721,167
1898	167,351	1,559	23,239	25,751	207,271	782,678
1899	162,613	1,606	17,570	23,990	196,938	781,873
1900 (a)	165,000	1,600	17,000	24,000	228,414	840,196

(a) Estimated in part.

The price of common pig lead in New York averaged for the year 1900, per pound avoirdupois, 4.37 cents. The two other principal lead markets are St. Louis and Chicago, where the lead is cheaper than in New York by the cost of transportation.

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CHAPTER II.

PROPERTIES* OF LEAD AND OF SOME OF ITS COMPOUNDS.

§ 4. LEAD.—Lead has a bluish-gray color; on a freshly cut surface it shows a considerable luster, but loses it quickly when exposed to atmospheric air. It does not crystallize readily. When it is cooled slowly, as in the Pattinson process (§ 98), bundles of small, imperfect octahedrons form. Also, when refined lead is poured at the correct temperature into a warm mold and is allowed to cool, fern-like crystalline aggregates appear at the surface. It is the heaviest of all base metals. Reich's figure, 11.37, as specific gravity for pure lead at 0° C. (water at 4° C. being unity), is the one that has been generally accepted. Roberts-Austen† gives as specific gravity of solid lead 11.40, of liquid lead 10.65 and 10.37.

The specific gravity of lead will vary slightly, according as it is cooled quickly or slowly, hammered or rolled. Commercial lead has a lower specific gravity than 11.37 on account of the impurities contained in it. Lead is very soft, especially when allowed to cool and solidify slowly. It is harder if cooled quickly and if it contains slight admixtures of other metals, such as copper, arsenic, antimony, zinc, etc. The grade of commercial lead is often approximately determined by the resistance it offers to scratching with the finger-nail, and the facility with which it makes a gray streak on paper. Lead is very malleable; it is rolled into sheets and hammered into foil. The fracture of lead is hackly when broken cold, columnar when hot. In the form of filings it becomes a solid mass if subjected to a pressure

* Condensed for the most part from Percy, "Metallurgy of Lead," London, 1870, pp. 8–93.
† "An Introduction to the Study of Metallurgy," London, 1894, p. 15.

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of 13 tons to the square inch, and liquefies at two and one-half times this pressure (Roberts -Austen).*

Lead is not sufficiently ductile to be drawn into fine wire; its tenacity, according to Karmarsch, is inferior to that of most ductile metals. It fuses at 325° C. (Le Chatelier),† boils at between 1450° and 1600° C. (Carnelly and Williams),‡ but cannot be distilled. It is somewhat volatile when heated to a cherry-red with access of air. The latent heat of lead is 5.369; the coefficient of cubical dilation for 1° C., 0.000089; the linear coefficient about one-third of the cubical. The specific heat between 10° and 100° C. is 0.0314; with silver as 100, the conductivity for heat at 12° C. is 8.5, and for electricity 10.7.

Lead undergoes no change in perfectly dry air nor in water that is free from air; its surface becomes, however, dull by oxidation when it is exposed to the atmospheric air on account of the moisture which it contains. Similarly, it is oxidized by water that is not free from air. If melted in contact with air it oxidizes and becomes covered with an iridescent pellicle, said to be the suboxide, Pb_2O; this gradually changes to the oxide, PbO, and if the heating to from 300° to 450° C. be prolonged sufficiently the red oxide, Pb_3O_4, is obtained. The other two oxides which lead forms are the sesquioxide, Pb_2O_3, and the peroxide, PbO_2.

The best solvent of lead is dilute nitric acid. Dilute hydrochloric and sulphuric acids have little or no action upon it; boiling concentrated hydrochloric acid and sulphuric acid of 66° B. dissolve it slowly. The researches of Lunge and Schmid ‖ on the action of sulphuric and nitric acid on lead of different degrees of purity gave important results. The following are the principal facts: The purer the lead the less will it be attacked by pure or nitrous sulphuric acid up to 200° C., the highest temperature employed under normal conditions in concentrating pans; above 200° C. the action becomes stronger and at 260° C. lead is suddenly dissolved completely, sulphur and sulphurous acid being formed. This sudden effect of the acid can be slightly retarded

**Engineering and Mining Journal*, Nov. 3, 1888; *Scientific American Supplement*, Nos. 675, 676, 677; "An Introduction to the Study of Metallurgy," p. 65.

†*Engineering and Mining Journal*, Oct. 11, 1890.

‡ *Journal of the Chemical Society*, xxxv., p. 563.

§ Lunge, *Engineering and Mining Journal*, Jan. 7, 14, 21, 1893; Schmid, "Ueber die Einwirkung von reiner, nitroser und rauchender Schwefelsäure und Saltpetersäure auf reines Blei und Legirungen von Blei mit Antimon und Kupfer," Bâle, 1892, W. Riehm, p. 134, tables xxiii., ill.; Lunge-Schmid, *Zeitschrift für Anorganische Chemie*, 1892, vol. ii., pp. 451, 642.

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by the addition of 1% antimony and prevented by the addition of 0.2% copper to the lead. Concentrated nitrous sulphuric acid acts at all temperatures more powerfully than pure sulphuric acid, and the effect is greater in the presence of air. Dilute nitrous sulphuric acid of a sp. gr. of 1.72 to 1.76 is not as powerful as the pure acid, although if the dilution be continued beyond this point the power increases again instead of diminishing. Junge* found that in the matter of corrosion by nitrous sulphuric acid a smooth or a rough surface made more difference than a slight variation in the chemical composition, the rough surface being the more readily attacked.

Organic acids—acetic, tartaric, and citric acids—attack lead in contact with air.

§ 5. Lead Oxide, PbO; 92.83% Pb.—Of the different oxides enumerated, this is the one that is metallurgically interesting and important. It is obtained on a large scale as massicot and litharge, which have different physical properties. Massicot, an amorphous yellow powder, is formed by heating lead on a flat hearth to a low red heat, removing the film of suboxide as fast as it forms, and oxidizing it to yellow oxide. If the temperature be raised to the melting point—that is, to a bright red heat—and the fused oxide cooled, it solidifies as crystalline litharge. Litharge is obtained on a large scale by cupelling argentiferous lead. It is soft and greasy to the touch and generally crystallizes in orthorhombic octahedrons. Ternier and Le Chatelier† have found tetragonal forms. While molten it is transparent and orange-colored; when cold it is opaque, and its color varies from yellow to red according to the rate at which it has cooled; quick cooling promotes the yellow, slow cooling the red color. Yellow litharge is produced on a large scale by allowing it to run from the furnace over an iron plate and chilling it with water, if necessary; it is thus obtained in small lumps. The red, flaky variety is formed by allowing the running litharge to collect in front of the furnace in cakes of from 1 to 1.5 tons in weight, and to cool slowly. The inner part of a cake will swell up and form flakes of red litharge; the outer and lower parts, having cooled quickly, will remain solid and have a yellow color. This swelling is

* *Freiberger Jahrbuch*, 1895, p. 11; *Zeitschrift für Angewandte Chemie*, 1895, p. 702; *Berg- und Hüttenmännische Zeitung*, 1896, p. 33.

† *Berg- und Hüttenmännische Zeitung*, 1895, p. 453.

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caused by the giving off of oxygen, which molten litharge absorbs. In solidifying quickly in small lumps the oxygen only makes the surface uneven; in cooling slowly in large lumps the outer solid crust obstructs the passage of the oxygen. This prevents the inner part from solidifying firmly, and causes instead the formation of loose flakes. The flakes and lumps are separated by sifting. Both varieties, when ground, have a reddish-yellow color. Litharge melts at 954° C. (Honsell);* it is a good conductor of electricity when molten. It is volatilized at a white heat. It is only slightly soluble in water (1 part in 12,000 parts), but is readily so in nitric and acetic acid.

Litharge is a strong base and quickly corrodes acid furnace material, with which it forms a silicate. It is an excellent flux, forming fusible compounds with oxides that are infusible alone. They do not always enter into chemical combination with it, but often are simply held in igneous solution by an excess of litharge. Thus fusible mixtures are formed with lime, baryta, magnesia, and alumina. The following table shows the proportion of litharge required to form fusible compounds with the principal metallic oxides:

1 part of	Cu_2O	CuO	ZnO	Fe_2O_3	Fe_3O_4	MnO	SnO_2	Sb_2O_3	SbO_2	As_2O_3	As_2O_5
Requires parts of litharge.	1.5	1.8	8	4	10	10	12 to 13	Fusible in all proportions.	5	0.4 to 0.8	0.25 to 1

Litharge readily gives up its oxygen, if heated in a stream of hydrogen or carbon monoxide, or if mixed with carbon. Being easily reduced to the metallic state, it forms an important oxidizing agent. This is seen by its behavior with S, Te, As, Sb, Sn, Bi, Cu, Zn, Fe. They become wholly or partly oxidized, and the oxides either volatilized or scorified by the surplus of litharge, a corresponding amount of lead, which combines with any unoxidized part, having been reduced.

§ 6. LEAD SILICATES.—Lead oxide and silica begin to combine at a temperature where the oxide becomes soft. In fact, it is disadvantageous to raise the temperature quickly if a silicate is to be formed. This can be seen in slag-roasting a galena ore to which fine sand has been added. If the time at which the roasted

* *Berg- und Hüttenmännische Zeitung*, 1886, p. 106.

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ore is pasty be shortened and the fusion urged, uncombined silica will be found with combined when the roast is decomposed in nitro-hydrochloric acid. All silicates that do not contain more silica than is required to form the tri-silicate, $Pb_2Si_3O_8$ ($2PbO.3SiO_2$), are fusible at a low temperature, forming a transparent vitreous mass; the singulo-silicate, Pb_2SiO_4 ($2PbO.SiO_2$), when fused is as fluid as water. If the proportion of silica be raised above that of the tri-silicate, the compound becomes less fusible; thus $2PbO.9SiO_2$ gives a porcelain-like mixture, and $PbO.18SiO_2$ fritts only to a porous mass. All fusible lead silicates are yellow; they become darker in proportion to the quantity of lead they contain. They change their color if they are contaminated with other metallic oxides, as can be seen if lead is slagged in a scorifier; *e.g.*, iron colors brown; copper, green; manganese, purple-black; nickel, brownish-yellow; cobalt, blue; tellurium, yellowish-red, the colors growing dark in proportion to the oxide added. Lead silicates are much used for glazing tiles, pottery, etc. Their behavior has been studied by Seger and Cramer.*

The lead from silicates is not readily liberated by the ordinary reducing agents. Sulphur decomposes the singulo-silicate to some extent, but it has less effect on the bi-silicate; iron sulphide throws down some lead, a double silicate of lead and iron being the result; carbon reduces from a bi-silicate part of the lead. In order to extract all the lead it must be first set free from its combination with silica by a basic flux; thus metallic iron decomposes all fusible lead silicates at a bright-red heat, providing enough is added to form a singulo-silicate.

The singulo-silicate and bi-silicate of lead are readily decomposed by nitric acid, the tri-silicate is not completely decomposed; the more acid the silicate the less soluble it is.

§ 7. Lead Sulphide, PbS; 86.6% Pb.—This occurs native as galena. It is formed artificially by heating lead and sulphur, or lead oxide with an excess of sulphur, or by reducing lead sulphate with carbonaceous matter, or by precipitation from a solution with hydrogen sulphide. The sulphide produced in the dry way has the same properties as the mineral.

The existence of subsulphides of lead (Pb_2S, Pb_4S) is denied by Percy, who shows that lead sulphide and lead can be melted

* *Thonindustrie-Zeitung*, 1893, p. 1294; *Berg- und Hüttenmännische Zeitung*, 1894, p. 11.

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together in all proportions, and that the properties of the resulting compound will resemble galena or lead according to the predominance of one or the other compound. Also, if such an apparent subsulphide be heated carefully, comparatively pure lead will eliquate and a residue of hard crystalline sulphide remain behind.

Rössler* found that lead sulphide dissolved in metallic lead crystallizes, as it cools, in cubes arranged in step form, resembling those which form in wall and hearth accretions of blast furnaces.

Galena is not as fusible as lead, melting, according to Lodin,† at 935° C., but it is very fluid when melted, and penetrates the firebrick of the furnaces in which it is treated; often a network of small veins of bright crystalline galena is found in furnace linings. When melted, galena begins to volatilize without being decomposed, if free oxygen be excluded. Lodin states that it is volatile below its melting point. On the walls of lead blast furnaces crystals of sublimed galena are of common occurrence. Lead sulphide is isomorphous with metallic sulphides, as Ag_2S, Cu_2S, ZnS, FeS. Such mixtures of sulphides are found in lead matte and copper matte obtained in smelting sulphide ores. With the electro-negative sulphides of antimony and arsenic it forms sulpho-salts. Quite a number of these occur as minerals; others can be artificially prepared in the dry and wet ways. Galena as well as matte is a good conductor of electricity.‡ Iron decomposes galena better than any other metal. For instance, copper that has a greater affinity for sulphur than iron decomposes lead sulphide only partially, as it alloys too readily with the liberated lead, and the cuprous sulphide formed combines with the remaining lead sulphide to form matte. Zinc decomposes galena partly, but the zinc sulphide formed is so refractory that a separation of the liberated lead is not effected; the result is a black, porous mass containing particles of lead and galena. The reaction that takes place between iron and galena is generally expressed thus:

$$PbS+Fe=FeS+Pb,$$

and forms the basis of what is called the precipitation or iron-

* *Chemical News*, lxxi., p. 293. † *Chemical News*, xxi. p. 293.

‡ Kiliani, *Berg- und Hüttenmännische Zeitung*, 1883, pp. 237, 366, 378.

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reduction process. In reality, however, the iron sulphide retains some undecomposed lead sulphide, and Nolte's* formula,

$$3PbS+3Fe=2Pb+(PbS+Fe_2S+FeS),$$

may be truer to the actual facts, although he presupposes the existence of an iron subsulphide. In decomposing galena in furnace-work by means of iron, enough has to be present to have 1Fe for 1PbS. If less is added, the resulting matte remains too rich in lead; if an excess is given, it is wasted. It may in fact be a disadvantage in decomposing argentiferous galena, since on account of the affinity the silver sulphide has for iron sulphide, more silver will go into the matte than can be accounted for by the amount of lead present. (See § 85.) In addition to having a correct amount of iron present to decompose galena, the temperature is of great importance; the higher it is, within reasonable limits, the better will be the decomposition. A basic ferrous silicate ($4\ FeO.SiO_2$) will decompose galena readily; the singulo-silicate ($2\ FeO.SiO_2$) shows little effect. In practice, the amount of iron that is in excess of that required to flux the silica will be available for the decomposition of the lead sulphide.

Lime as well as baryta has a decomposing action on galena. If air has access, the following reaction takes place (Rivot):†

$$4PbS+4CaO=3CaS+CaSO_4+4Pb.$$

If the air be excluded and carbon present, the following occurs (Berthier):‡

$$2PbS+CaO+C=Pb+(PbS.CaS)+CO.$$

Hannay§ believes that there are two volatile compounds of lead sulphide of the formulæ $PbS.H_2O$ and $PbS.SO_2$, but other evidence‖ points to their non-existence.

§ 8. Lead Sulphate, $PbSO_4$; 68.3% Pb.—This occurs as anglesite. It is formed in roasting lead sulphide (§ 9). Of all metallic sulphates it is the only one that is not decomposed upon ignition at a bright-red heat; it softens at a white heat and loses

* *Berg- und Hüttenmännische Zeitung*, 1860, p. 165.

† "Traité de métallurgie," Paris, 1872, vol. ii., p. 42.

‡ "Traité des essais par la voie sèche," Liege, 1887, vol ii., p. 580.

§ *Institution of Mining and Metallurgy*, London, ii., p. 171.

‖ Roberts-Austen, *Chemical News*, lxx., p. 43; Jenkins and Smith, *Journal of the Chemical Society*, London, 1897, lxxi., and lxxii., p. 666, and *Engineering and Mining Journal*, June 19, 1897.

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some of its sulphur trioxide, forming a basic salt. Silica readily decomposes the sulphate, forming a lead silicate, while the sulphur trioxide is driven off, being split into sulphur dioxide and oxygen. In this way lead sulphate, obtained in roasting a silicious galena, is decomposed. The operation goes by the name of slag-roasting. This decomposition of lead sulphate is commonly expressed by

$$2PbSO_4+SiO_2=Pb_2SiO_4+2SO_3,$$
$$2SO_3=2SO_2+2O.$$

While it is not absolutely necessary to follow these proportions to effect the decomposition (see § 6), the usual practice (see § 55) does so pretty closely, as the singulo-silicate is readily fusible at a low temperature and very fluid, thus assisting the fusion and solution of the less fusible components of the charge. Carbon, if present in sufficient quantity, reduces the sulphate completely to sulphide at a dark-red heat:

$$PbSO_4+2C=PbS+2CO_2.$$

If there is not enough carbon present, only part of the sulphate will be reduced:

$$2PbSO_4+2C=PbSO_4+PbS+2CO_2, \text{ and}$$
$$4PbSO_4+2C=3PbSO_4+PbS+2CO_2.$$

At a cherry-red heat the resulting sulphate and sulphide will react upon each other, as shown in the next paragraph.

Lead sulphate is only slightly soluble in water and dilute sulphuric acid, more so in nitric acid and solutions of nitrates; it is soluble to some extent in sodium hyposulphite, the solubility increasing with the concentration and the temperature of the solvent (Stetefeldt);* it is readily soluble in calcium chloride. Lead sulphate is a poor conductor of electricity (Kiliani).†

§ 9. Roasting of Lead Sulphide; Reactions Between Lead Sulphide, Lead Sulphate, and Lead Oxide.—The roasting of galena (see § 55) and making the products react upon still undecomposed sulphide at an elevated temperature is of special interest, since one important lead-smelting process, the roasting and reaction process, also called the air-reduction process, is based upon

* "The Lixiviation of Silver Ores with Hyposulphite Solutions," New York and San Francisco, 1895, p. 25.

† *Berg- und Hüttenmännische Zeitung*, 1883, p. 237.

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it. If galena is ground fine and roasted* carefully at a temperature sufficiently low to prevent its becoming pasty, it will first be converted into oxide (perhaps only into suboxide) and sulphur dioxide. Lead sulphide does not oxidize readily, and the dioxide will therefore form slowly. As we require a low temperature, only part of the dioxide combines with the oxygen of the air and forms the trioxide by contact, and this combines with the lead oxide, forming sulphate. If lead suboxide is present the sulphur trioxide converts this first to oxide. Rammelsberg† suggests that some lead sulphide is directly oxidized to lead sulphate without passing through the stage of oxide. An experiment with pure galena gave to Plattner approximately the proportion $5PbO : 2PbSO_4$; on roasting a galena from Bleiberg (Carinthia) which contained a small amount of blende and pyrite this changed to $PbO : PbSO_4$; with 50% of pyrite added the ratio was $2PbO : 3PbSO_4$.

This shows that the relation of lead oxide and sulphate in roasted galena depends on the presence of other sulphides. It is generally accepted that slow roasting at a low temperature produces more sulphate than if the operation be carried on quickly at a higher temperature, which may perhaps be expressed by

$$2PbS+7O=PbO+PbSO_4+SO_2 \text{ (slow roasting)},$$
$$3PbS+10O=2PbO+PbSO_4+2SO_2 \text{ (quick roasting)}.$$

But, according to Rammelsburg,‡ this is not definitely settled.

If lead sulphide be heated to a strong red heat with lead oxide or sulphate, the following reactions will take place:

(1) $PbS+2PbO=Pb_3+SO_2$.
(2) $PbS+3PbO=Pb_3+PbO+SO_2$.
(3) $2PbS+2PbO=Pb_3+PbS+SO_2$.
(4) $PbS+PbSO_4=Pb_2+2SO_2$.
(5) $PbS+2PbSO_4=Pb+2PbO+3SO_2$.
(6) $PbS+3PbSO_4=4PbO+4SO_2$.

Lodin§ found that reaction (1) begins at 720° C. and reaction (4) at 670° C., continuing up to 820° C.

* Plattner, "Die metallurgischen Röstprocesse, theoretisch betrachtet," Freiberg, 1856, p. 145.

† Percy-Rammelsberg, "Die Metallurgie des Bleies," Brunswick, 1872, p. 39.

‡ *Op. cit.*, p. 40.

§ *Comptes Rendus de l'Académie des Sciences*, vol. cxx., pp. 1164-1167; *Chemisches Centralblatt*, 1895, ii., p. 15.

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These equations show that with correct proportions (1 and 4) all the lead is reduced by the sulphur, which combines with the oxygen to dioxide; if we have an excess of lead oxide (2), or of sulphide (3), it remains unaltered. We find something similar to be the case with equations (5) and (6). With too much lead sulphate we retain some or all of the lead as oxide; with a surplus of sulphide it would remain unchanged.

§ 10. LEAD CARBONATE, $PbCO_3$; 77.5% Pb.—This occurs as cerussite. It is a poor conductor of electricity.* The white lead of commerce is a basic carbonate. Lead carbonate is readily decomposed at a very low temperature (200° C.) into oxide and carbon dioxide.

§ 11. THE LEAD OF COMMERCE: ITS IMPURITIES AND THEIR EFFECT.—In the market we find three kinds of lead: undesilverized lead, desilverized lead, and antimonial lead. The first comes from the non-argentiferous ores of the Mississippi Valley, the second from the refining works which desilverize argentiferous lead (Part III.). The third is a by-product of the second. The two soft leads are manufactured into sheet-lead and lead pipe; they are used for making alloys, for corroding, and for other chemical purposes that require a good grade. The hard lead is used to some extent in making type-metal, bearings, etc., but generally the makers of these alloys prefer to use soft lead and antimony, and the greater part of the antimonial lead is used for purposes for which an inferior lead will suffice, as, for instance, coffin-cases, selling in the market about ⅛c. per lb. less than ordinary desilverized lead. (See § 12.)

According to Caswell† the lead of this country is absorbed by the different manufactures as follows: Paint (white lead, litharge, etc.), 35%; lead pipe, 20%; sheet lead, 8%; shot and bullets, 9%; solder, 5%; tamping, filling knobs, trimmings, etc., 23%—total, 100%.

In the subjoined table are given the analyses of the principal American brands of lead; some well-known European makes have been added for the sake of comparison.

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* Kiliani, *Berg- und Hüttenmännische Zeitung*, 1883, p. 287.
† *Iron Age*, Jan. 2, 1896.

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	Harz Mountains.			Pribram.	Pennsylvania Lead Co.	Cons. Kansas City Smelting and Refining Co.		Pueblo Smelt. and Refin'g Co.
	Reduced from Litharge	Pattinson Process.	Parkes Process.	Luce-Rozan Process.	Parkes Process.	Parkes Process.		Parkes Process.
Cu	0.0600	0.0150	0.00080	0.0024	0.00007	0.00022	Trace.	None.
Ag	0.0028	0.0022	0.00085	0.0018	0.00042	0.00020	0.0004	0.000514
Bi	Trace.	0.0006	.....	0.0023	.....	0.00308	.....	0.001500
Cd	.....	.....	0.00088	.....	.....	Trace.	.....	.....
Sn	.....	.....	.....	.....	.....	.....	.....	.....
As	.....	.....	.....	.....	.....	Trace.	Trace.	0.000120
Sb	0.1340	0.0100	0.00796	0.0056	0.00051	0.00127	0.0004	0.002298
Ni	0.0050	0.0010	.....	0.0011	.....	Trace.	.....	.....
Co	.....	.....	.....	.....	.....	.....	.....	.....
Fe	0.0080	0.0040	0.00100	0.0017	Trace.	0.00178	0.0006	0.00150
Zn	0.0040	0.0010	0.00092	0.0010	0.00088	0.00075	0.0013	0.00250
Mn	.....	.....	.....	.....	.....	0.00021	.....	.....
S	.....	.....	.....	.....	0.00018	.....	.....	.....
Pb	99.7912	99.9662	99.98480	99.9841	99.99644	99.99249	99.9963	99.98037
Reference.	(a)	(a)	(b)	(c)	(d)	(e)	(e)	(f)

(a) *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xvii., p. 205. (b) Private Notes, 1890. (c) *Oesterreichische Zeitschrift für Berg- und Hütten-Wesen*, 1890, p. 497. (d) "Transactions of American Institute of Mining Engineers," iii., p. 322. (e) *Engineering and Mining Journal*, July 14, 1892. (f) Private notes.

1. *Copper.*—Copper and lead do not readily form homogeneous alloys. In order to obtain these, the two metals have to be melted together beyond the fusing point of copper and then chilled quickly. If such an alloy be heated gradually to the melting point of lead, it is possible to separate part of the lead from the copper. This will retain as minimum 0.08% copper.* The copper retaining some lead will remain behind as a porous mass. In order to remove the copper from the lead an addition of zinc is necessary, which will extract it to the practical limit (Parkes Process, § 101). The percentage of copper in commercial lead does not interfere with the rolling and other mechanical process. According to Lunge and Schmid,† lead with 0.1% copper shows the same behavior as pure lead with cold sulphuric acid, so does lead with 0.2% copper up to 100° C. At 200° C. lead with from 0.1 to 0.2% copper is almost as much attacked as pure lead is, and more than it is by nitrous sulphuric acid. Junge ‡ says that in concentrating pans Parkes lead has been observed to be more readily attacked than Pattinson lead, and he attributes this to the fact that there is less copper in the Parkes lead, which making the melting point lower increases the liability of being attacked by the acid. If used for corroding or for making flint-glass the percentage of copper ought not to exceed

* See Softening of Base Bullion, § 104. † *Op. cit.* ‡ *Op. cit.*

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0.0014% (Hampe).* According to Junge the lowest permissible amount is 0.003%.

2. *Silver.*—What is said about copper in regard to mechanical treatment holds good for silver. Small quantities of silver protect lead against sulphuric acid. Baker† says that 1.70 oz. of silver per ton gives white lead a reddish tinge, while this is not the case with 0.15 oz. per ton. Landsberg ‡ gives as minimum 1.03 oz. for corroding lead; Junge, 0.17 oz.

3. *Bismuth.*—A percentage of 0.118 to 0.352% bismuth makes lead hard,§ somewhat crystalline, and more readily fusible (Hampe). Junge rolled without difficulty lead containing 2% bismuth into a sheet 10 ft. long, 6 ft. wide and $\frac{1}{64}$ in. thick, and Burgraf says that lead with 1.920% bismuth can be rolled into sheets as well as refined lead practically free from bismuth.

According to Napier‖ and Bauer,¶ 0.1% bismuth protects lead somewhat from sulphuric acid at 20° C., but not at 100° C., while Junge** exposed strips of lead with 0.20% bismuth to sulphuric acid of 60° B. at the temperature of the sulphuric acid chamber and found no sensible change. Lunge and Schmid†† state that lead containing bismuth is suddenly dissolved by concentrated sulphuric acid at a temperature below 260° C., and that an addition of 0.02% copper counteracts this to some extent. According to Hampe, from 0.0045 to 0.0075% bismuth has no effect on white lead, while Junge says that 0.003% makes itself felt. Refined lead containing over 0.04% bismuth‡‡ is no longer classed in this country as corroding lead. Endemann§§ states that bismuth favors the corrosion of lead, a small black residue remaining behind containing metallic bismuth. From 0.0045 to 0.0075% bismuth has no effect on white lead (Hampe). The only means of removing bismuth from lead is by the Pattinson crystallization process (§ 98).

4. *Cadmium.*—This occurs only in traces. It protects lead from the action of sulphuric acid.

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* *Zeitschrift für Berg-, Hütten- u. Salinen-Wesen in Preussen*, xviii., p. 209.

† Dingler, *Polytechnisches Journal*, clxxiii., p. 119.

‡ *Wagner's Jahresberichte*, 1875, p. 596.

§ Plattner, *Berg- und Hüttenmännische Zeitung*, 1869, p. 116.

‖ *Chemical News*, 1880 p. 314; *School of Mines Quarterly*, vii., p. 97.

¶ *Berichte der deutschen chemischen Gesellschaft*, 1875, p. 40; *School of Mines Quarterly*, vii., p. 117. ** *Op. cit.* †† *Op. cit.*

‡‡ Eakins, *Proceedings of the Colorado Scientific Society*, Feb. 14, 1895.

§§ *American Chemist*, 1876, vi., p. 457, and *Wagner's Jahresberichte*, 1877, p. 422.

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5. *Tin.*—Tin makes lead light gray, hard, and increases its fusibility. It is uncommon in market leads. Lead containing it is more affected by sulphuric acid than pure lead (Napier, Bauer).* The effect in corroding has not been studied. It is removed by heating the lead to a bright-red heat with access of air; part of the tin collecting on the surface as oxide is first drawn off as a powder, and the rest as a slag consisting of stannic oxide and lead oxide. (Softening of Base Bullion, § 104.)

6. *Antimony* (*Arsenic*).—Even small quantities of antimony give lead a grayish-white color, and make it harder and less malleable than ordinary lead. A bar of lead containing some antimony will show, especially in the center, an uneven, moss-like surface. Hampe† finds that 0.005% antimony does not harden lead; Heeren‡ states that 0.25% makes lead hard, but that it is still malleable. Lead with 0.1% antimony is not so easily attacked by cold sulphuric acid as pure lead, but more easily by hot acid. Lunge and Schmid (*loc. cit.*) substantiate this. They found that an addition of 0.2% antimony did no harm with cold sulphuric acid, appearing rather to be beneficial; but lead with over 0.2% antimony was more affected than soft lead, and the discrepancy increases with a higher temperature to an enormous degree.

For corroding, lead may not contain over 0.005% antimony (Hampe, Landsberg). Junge (*loc. cit.*) finds that 0.05% antimony retards the corrosion, but has no effect on the color of the white lead, while 0.1% has a decidedly bad influence. Antimony and arsenic are removed, if the lead is heated to a bright-red heat with access of air, as antimoniate and arseniate of lead ($Pb_3 2Sb(As)O_4$), slagged by an excess of litharge. In fact, if tin, arsenic, and antimony are present they will be oxidized in the order named, and can be worked up separately. (Softening of Base Bullion, § 104.)

7. *Nickel* (*Cobalt*).—These rarely occur in market lead. Berthier§ produced a malleable alloy containing from 0.4 to 0.5% nickel. Mrázek‖ says that from 1 to 2% antimony favors the entrance of nickel and cobalt into the lead, but they rise to the

* See Bismuth above, p. 22, notes 7 and 8. † *Loc. cit.*
‡ Percy-Rammelsberg, "Die Metallurgie des Bleies," p. 49.
§ "Traité des essais par la voie sèche," Liege, 1847, ii., p. 595.
‖ *Berg- und Hüttenmännische Zeitung*, 1864, p. 315.

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surface when the furnace lead is melted down slowly, and can then be easily skimmed off.

8. *Iron.*—Alloys of lead and iron form under special conditions, but market lead contains in maximo 0.07% iron (Reich),* which does not have any effect on the softness and malleability. Corroding lead ought not to contain over 0.003% iron (Landsberg).†

9. *Zinc.*—Zinc and lead can be melted together in varying proportions, but they separate again in part on cooling. According to Rössler and Edelmann,‡ the amount of zinc that lead will retain depends on the temperature of the lead. The subjoined table illustrates this.

Degrees Centigrade	400	500	600	700
Per cent. of zinc retained	0.6 to 0.8	0.9 to 1.3	1.5 to 2.3	3.0

Zinc gives lead a silvery color and makes it so hard that it cannot be rolled; cold and hot sulphuric acid attack it readily. Corroding lead should not contain over 0.003% (Landsberg). Zinc is removed from lead by heating it to a bright-red, and oxidizing it by admitting air, introducing steam, etc. (See Dezincification of Desilverized Lead, § 112.)

10. *Manganese.*—This is present only in very small amounts and has no practical importance.

11. *Aluminum.*§—Aluminum and lead show very little tendency to combine. When melted down together and allowed to cool slowly they separate easily; lead being more readily fusible will remain liquid while the aluminum floating on top will solidify. The lead will have taken up some aluminum and the aluminum some lead. Richards found 0.5% aluminum in lead. Antimony seems to favor the combination of lead with aluminum. The aluminum taken up is, however, readily removed, as in using aluminized zinc for desilverizing argentiferous lead in the Rössler-Edelmann process (see § 107) no aluminum appears in the refined lead.

§ 12. *Lead Alloys.*—Lead forms alloys with a number of metals,

* *Berg- und Hüttenmännische Zeitung*, 1860, pp. 28, 284.

† *Loc. cit.*

‡ *Berg- und Hüttenmännische Zeitung*, 1890, p. 245; *Engineering and Mining Journal*, Nov. 15, 1890.

§ Richards, "Aluminium," Philadelphia, 1896, p. 503.

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as shown in the previous section. Some of the leading industrial alloys are recorded below:*

Alloy.	Pb.	Sb.	Sn.	Cu.	As.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Type metal	55.0	30.0	15.0	.....	.....
Type metal	60.0	25.0	15.0	.....	.....
Type metal	70.0	18.0	10.0	2	.....
Type metal	80.0	20.0	.....	.....	.....
Type metal	82.0	14.8	3.2	.....	.....
Bearing metal	60.0	20.0	20.0	.....	.....
Bearing metal	77.8	16.3	5.9	.....	.....
Shot metal	99.6	.....	.....	.....	0.20–0.35
Solder, fine	33.3	.....	66.6	.....	.....
Solder, tin	50.0	.....	50.0	.....	.....
Solder, plumbers'	66.6	.....	33.3	.....	.....

* Ledebur, "Verarbeitung der Metalle auf chemisch-physikalischem Wege," Brunswick, 1882, pp. 5, 100; Kerl-Stohmann, "Handbuch der technischen Chemie," Brunswick 189–, vol. i., p. 1643; Behrens, "Das mikroskopische Gefüge der Metalle und Legirungen," Hamburg and Leipsic, 1894, pp. 60, 61; Brannt, "Metallic Alloys," Philadelphia, 1897, p. 360; Roberts-Austen, "An Introduction to the Study of Metallurgy," London, 1894, pp. 106, 107.

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CHAPTER III.

LEAD ORES.

§ 13. Introductory Remarks.—Many minerals contain lead, but only two or three are found in sufficient quantity to be sources of lead. The ores of lead are divided into two classes: Sulphide Ores (galena) and Oxidized Ores (anglesite, cerussite, etc.), commonly called Carbonate Ores. Both are made more or less impure by other metallic compounds and vein matter.

§ 14. Galena, PbS; 86.6% Pb, 13.4% S.—This mineral is found well crystallized in cubes, sometimes also in isometric octahedrons. Crystals are not so often found isolated as in irregular bunches. It occurs also in coarsely crystalline to fine granular varieties; crypto-crystalline galena is rare. Galena is found in most of the geological formations, but especially in the Silurian, Carboniferous, and Triassic. It is usually mixed with more or less vein matter, such as limestone, dolomite, sandstone, quartz, earthy carbonates, barite, clay-slate, granite, gneiss, etc., which often have to be removed by a mechanical washing process before sending the mineral to the smelter. The mine which works on the lowest grade galena ore is probably that at Mechernich, Rhenish Prussia, where small nodules of galena the size of a pea occur in a soft Triassic sandstone, the grains of which, about the size of millet, are slightly cemented by a clay or lime cement. The ore contains only 2.5% of galena, and this runs only about 6 oz. silver per ton.

The following table shows the rock and formation in which some well-known galena deposits occur and the tenor of the ore before and after dressing:

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Locality.	Nature of Rock.	Geological Formation.	Raw Ore: per cent lead.	Dressed Ore.		Reference.
				Per Cent. Lead.	Oz. Silver Per Ton.	
Mineral Point, Wis.	Dolomite	Lower Silurian	....	.....	3.00	1
Rockville, Wis.	Dolomite	Lower Silurian	....	.....	0.30	1
Granby, Mo.	Dolomite	Sub-Carboniferous.	....	84	1.25	1
St. Joseph, Mo.	Dolomite	Lower Silurian	7	70	.....	2
North of England.	Limestone	Carboniferous	8.5	70–77	8.00	3
Bleiberg, Carinthia	Limestone	Triassic	8	71	0.05	4
Pribram, Bohemia	Graywack	Lower Silurian	20	37–38	76.50	5
Freiberg, Saxony	Gneiss	Archæan	3	18–70	17–88	6
Tarnowitz, Silesia	Dolomite	Triassic	6	75.5	13.50	7
Upper Harz, Prussia	Graywack slate	Sub-Carboniferous.	9	64	25.00	8
Mechernich, Prussia	Sandstone	Triassic	2	56–60	3–4	9
Kellogg, Idaho	.....	.....	11	60	30.00	10

1. "Geological Survey of Wisconsin," 1873–79, iv., p. 382.
2. Desloge, "Transactions of American Institute of Mining Engineers," xviii., p. 202.
3. Hunt, "British Mining," London, 1884, p. 899; and Phillips, "Elements of Metallurgy," Philadelphia, 1887, p. 566.
4. *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1890, p. 286.
5. *Ibid.*, 1888, p. 567; and *Oesterreichisches Jahrbuch*, xxxix., p. 10.
6. "Freiberg's Berg- und Hüttenwesen," Freiberg, 1883.
7. *Zeitschrift für Berg- Hütten- und Salinen-Wesen in Preussen*, xxxii., p. 202.
8. *Ibid.*, xxx., p. 131 and Private notes, 1890.
9. "Bergbau und Hüttenbetrieb von Mechernich," Cologne, 1886.
10. Private notes.

Galena often occurs in a very pure state, but it is more generally contaminated with other metallic sulphides. These are either pyrite, arsenopyrite, chalcopyrite, blende, bournonite, etc., as associated minerals; or silver, copper, zinc, iron, nickel, etc., forming isomorphous compounds with the lead sulphide. The associated minerals can usually be mechanically separated from the galena, but not always. Sometimes the admixture is too intimate, causing trouble and loss in the metallurgical treatment.

Galena is almost always argentiferous. The silver is rarely present in the native state; commonly it appears as isomorphous silver sulphide, or as a finely disseminated silver mineral. The difference of form is important in connection with wet concentration.* If the silver occurs as isomorphous sulphide, replacing part of the lead sulphide, the loss in concentration will correspond to the percentage of lead in the tailings; if as associated mineral (*e.g.*, tetrahedrite), it will be very great, as this mineral, being very brittle, is readily crushed to a fine powder, and, being also lighter than galena, is carried off on the water. If the dark scum that is often seen floating on the water of jigs, where argen-

* Raymond, *Engineering and Mining Journal*, Feb. 11, 1882.

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tiferous galena is concentrated, be assayed, the main source of loss in silver appears.

The tenor of silver in galena ores varies a great deal. The galena from Bleiberg, Carinthia, with 0.05 oz. silver per ton, represents probably the lowest amount, and occasional specimens from Idaho and Schemnitz, Hungary, with 2,042 oz. the highest. The amount of silver contained in galena is often said to be dependent on the inclosing rock; thus crystalline rocks would be favorable to a high percentage of silver, and unchanged sedimentary strata unfavorable. While many occurrences of galena appear to sustain this rule, others again disprove it, so that it is not of general application.

It has often been said, and may sometimes still be heard, that coarse-grained galena is poor in silver, while fine granular varieties give higher assays, but Malaguti and Durocher* disproved it 40 years ago. The minerals oftenest associated with galena—such as pyrite, blende, etc.—do not generally contain as much silver as the galena.

Percy† states that gold is as invariably present in galena as silver, but it does not often occur in appreciable quantities.

§ 15. ANGLESITE, $PbSO_4$; PbO, 73.6; SO_3, 26.4; Pb, 68.3. CERUSSITE, $PbCO_3$; PbO, 83.5; CO_2, 16.5; Pb, 77.5.—Carbonate ores, using that term in a general sense as embracing all oxidized lead ores, occur often in the form of a sand or an earth, bearing the name of sand or soft carbonates. In other cases the particles of carbonate are cemented together by clay, iron, manganese, or silicious matter, forming compact amorphous or crystalline lumps called hard carbonates.

The minerals are seldom found as originally deposited; the form and composition are more apt to have been caused by chemical changes that have taken place since the galena from which they resulted was deposited. The sulphate usually formed by the oxidation of the sulphide is in most cases the compound from which the carbonate has resulted by the action of circulating waters holding alkaline or earthly carbonates in solution.

Anglesite as an ore is rare, as it is not often that galena is exposed only to the oxidizing action of air. It often occurs, however, with cerussite, and represents the transition between

* *Annales des Mines*, Fourth Series, 1850, xvii., p. 85.
† "Metallurgy of Lead," p. 96.

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the sulphide and the carbonate. This is further illustrated by pieces of galena found in carbonate ores, which contain in the interior anglesite and the surface of which is changed to cerussite. Thus these two minerals will be always found near the outcrop of galena deposits. To what extent the decomposition has progressed depends on local circumstances.

The richness in lead of carbonate ores varies a great deal. If galena alone has been oxidized, the ores are likely to be rich; if the decomposing action attacked also the country rock, this may so contaminate the ore as to reduce the percentage of lead below the limit where it pays to treat the ore. The grade of carbonate ore is not so often raised by wet concentration as that of galena ores, as the losses in lead, and especially in silver, which passes off in the slimes, are almost unavoidable. In some instances the carbonate ore has been first leached with sodium hyposulphite to remove as much silver as possible, and then the lead concentrated in the wet way to a rich product. This used to be done at the Old Telegraph* Mine, Utah. Another method that has been tried is Krom's† system of dry concentration, the result being a high grade smelting ore, and tailings and dust, to be treated in the wet way. The following table shows some very pure carbonate ores from Missouri resulting from corresponding pure galena, and others from Colorado contaminated with country rock.

Locality.	Per Cent. Lead.	Ounces Silver per ton.	Chemist.	Reference.
South West, Mo.....	72	.....	Chauvenet.....	Broadhead, "Geological Survey of Missouri," 1874, p. 710.
Granby, Mo.....	65	.....	Williams.....	"Trans. A. I. M. E.," v., 315.
Leadville, Colo.....	38	25	Ricketts.....	"Trans. A. I. M. E.," xiv., 397.
Leadville, Colo.....	21	65	Rolker.....	"Trans. A. I. M. E.," xiv., 397.
Red Mountain, Colo.....	17	128	Kedzie.....	"Trans. A. I. M. E.," xvi., p. 561.

The associated minerals undergo a process of oxidation with the galena and are generally found again in part in the carbonate ore, although, being more soluble than the lead sulphate, they

* "Tenth Census of the United States," 1880, vol. xiii., p. 415.

† Krom, "Commissioner Raymond's Report," 1876, p. 419; "Transactions of American Institute of Mining Engineers," xiv., p. 497; *Engineering and Mining Journal*, Aug. 14, Sept. 25, Oct. 23, 1886; Stetefeldt, *Engineering and Mining Journal*, Oct. 28, 1876; May 2, 1885; "Transactions of American Institute of Mining Engineers," xv., p. 355; Furman, *School of Mines Quarterly*, iii., p. 127: Newberry, *Ibid.*, iv., p. 1; Heard, *Engineering and Mining Journal*, July 3, Sept. 11, 1886; Dry Recorder, *Ibid.*, Sept. 4, 1886; Hollister, "Transactions of American Institute of Mining Engineers," xvi., p. 1; Sickel, *Berg- und Hüttenmännische Zeitung*, 1885, p. 313; Blömecke, *Berg- und Hüttenmännische Zeitung*, 1886, pp. 485, 501, 514.

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may be carried away entirely. The silver in oxidized ores is present mostly in the form of chloride, although it also occurs in its original form as sulphide and antimonide. There is less likelihood of a uniform ratio between lead and silver in a carbonate ore than there is in the sulphide ore, as lead sulphate and carbonate behave differently with solvents from silver sulphide, chloride, and antimonide. Thus enrichment and impoverishment both in lead and silver can be easily accounted for.

In carbonate ores the gold occurs probably in the native state.

§ 16. OTHER LEAD MINERALS.—The following six oxidized lead minerals occur often in carbonate deposits, but not in sufficient amounts to constitute an ore:

Pyromorphite, $PbCl_2+3\ Pb_3P_2O_8$; 76.4 Pb.—Calcium fluoride often replaces in part the lead chloride; calcium, the lead combined with phosphoric acid, and arsenic acid, the phosphoric acid.

Mimetite, $PbCl_2+3\ Pb_3As_2O_8$; 69.6 Pb. In the lead arseniate the lead is sometimes in part replaced by calcium, and the arsenic usually in part by phosphoric acid.

Vanadinite, $PbCl_2+3Pb_3V_2O_8$; 65 Pb.

Crocoite, $PbCrO_4$; 63.9 Pb.

Wulfenite, $PbMO_4$; 57 Pb.

Stolzite, $PbWoO_4$; 45.4 Pb.

There might still be mentioned about 20 lead-bearing sulpharsenites, sulphantimonites, and sulphobismuthites, sulphoarsenates, sulphantimonates, etc., which occur in lead deposits, but they are only mineralogical curiosities.*

* Dana "System of Mineralogy," New York, 1892, pp. 109-152.

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CHAPTER IV.

DISTRIBUTION OF LEAD ORES.

§ 17. Lead Ores of the United States.*—Lead ores occur in many parts of the world. The mines of Spain, Germany, and England furnish the bulk of the European product, but it is not the present purpose to deal with that branch of the subject.

The occurrence of lead ores in the United States is best discussed under four heads: I. The Atlantic coast: II. the Mississippi Valley; III. the Rocky Mountains; IV. the Pacific.

I.—Lead Ores of the Atlantic Coast.

§ 18. Lead Ores of the Atlantic Coast.—The lead ores of the Atlantic coast occur in New York, New England, Virginia, North Carolina, and Tennessee. Those of New York and New England, which were worked in former times, are practically abandoned now, and those of the Southern States are worked only on a small scale and spasmodically.

The Archæan gneiss of New York is traversed by veins of galena, which, being free from zinc and iron, have a gangue consisting mainly of calcspar. The Rossie mines were worked at intervals for over 50 years, but have been idle since 1864.

In the New England States, galena, more or less argentiferous and associated with other metallic sulphides, occurs irregularly distributed in segregated veins in highly metamorphosed palæozoic rocks.

Virginia has some deposits of galena and blende with carbonate and silicate of zinc; the lead, however, is subordinate to the zinc.

* Whitney, "Metallic Wealth of the United States," Philadelphia, 1854, p. 382; Ingalls, "The Distribution and Production of Lead," in "The Mineral Industry," 1893, pp. 381–420; Kemp, "The Ore Deposits of the United States," New York, 1895; Winslow, "Lead and Zinc Deposits," vols. vi. and vii. of "Missouri Geological Survey," 1894.

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II.—Lead Ores of the Mississippi Valley.

This heading covers two divisions: the lead region of the Upper Mississippi Valley and that of Missouri. They have many features in common and are often discussed together; there are, however, so many differences in the mode of occurrence of the galena, in the associated minerals, and even in the geological horizon, that it is advisable to keep them separate.

§ 19. The Upper Mississippi Valley.—In the southwestern part of Wisconsin are the important lead deposits of the Upper Mississippi Valley, which extend a very small way into the adjoining States of Iowa and Illinois. They are principally centered around Mineral Point and Platteville, Wis.; Galena, Ill., and Dubuque, Iowa. About 90% of the lead produced comes from Wisconsin.

The ore is a non-argentiferous galena; it occurs in wholly undisturbed dolomitic limestone of the Trenton period in vertical crevices, flat crevices, or as an impregnation. The vertical crevices (gash veins) are either thin seams in the rock, a few inches thick by several hundred feet long, extending downward for from 20 to 40 (occasionally 100) ft., filled up solid with coarse galena; or more commonly they expand in an irregular way, inclosing particles of rock, which are cemented together by galena. If disintegration has taken place, caves are formed in which the galena is found distributed among loose masses of rock, calcareous sand, clay, ocher, etc. The horizontal crevices (flat openings, flat sheets) are either thin seams of compact ore along the bedding planes of the rock, or, if the rock has been disintegrated, have been enlarged, and the galena is found in the same form as in the caves of vertical crevices. Combinations of vertical and horizontal crevices (flats and pitches) increase the size of the deposits. Impregnations of certain strata of dolomite occur, but they are not frequent.

The galena (mineral) from the upper beds is pure and rich. As depth is gained, the associated minerals—pyrite, blende—increase in quantity and often predominate over the galena. Chalcopyrite is scarce, and is found more with pyrite and blende than with galena. Secondary minerals are not of frequent occurrence. Calcite and barite occur in the lower beds. The absence of nickel, cobalt, and arsenic is to be noted.

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§ 20. The Mines of Missouri.—In both the Southeastern and the Southwestern districts the ore is a coarsely crystalline galena, practically free from silver.

In the Southeastern district, represented by the Bonne Terre and La Motte mines, galena occurs disseminated through strata of dolomitic limestone of the Lower Silurian period, lying almost horizontally. The thickness of the lead-bearing "Third Magnesian Limestone" varies usually from 2 to 6 ft., although it sometimes goes much higher. The ore as mined at Bonne Terre* runs about 7% lead and is concentrated to a product of 70% lead. Associated with the galena occurs pyrite containing nickel and cobalt. Chalcopyrite, as well as nickel and cobalt sulphides with traces of arsenic, is found more at Mine La Motte than at the Bonne Terre mines. The absence of blende is to be noted. The pyrite-bearing galena ores are concentrated (separately from the pure galena) to a product called "sulphide," which has the following composition:

Locality.	Lead.	Iron.	Nickel and Cobalt.	Sulphur.	Insol.	Chemist.
Bonne Terre, 1884.	21.86	16.21	0.61	.....	.....	Setz.
Mine La Motte, 1881	17.37	.....	4.77	.....	.....	Neill.
Mine La Motte, 1881	13.34	44.84	4.07	20.37	3.58	Neill.

The dolomitic limestone in which the ores occur contains about 3% silica; a marked feature is the presence of barite in places where the usually disseminated mineral has been concentrated to small sheets.

The ores of the Central Lead Region, lying between the Osage and Missouri Rivers, also occur in dolomite of the Lower Silurian period; they resemble those of Wisconsin, but are of no special importance.

In the Southwestern Region, which reaches into Kansas, Arkansas, and the Indian Territory, lead and zinc ores are worked very extensively at present, especially around Joplin. The galena occurs in dolomitic limestone of the Sub-Carboniferous period, containing layers of chert and bituminous matter. When broken it often emits a bituminous odor. It occurs in single or loosely aggregated crystals, also in crystalline masses of small dimen-

* Munroe, "Transactions of American Institute of Mining Engineers;" Desloge, *Ibid.* xviii., 263; Kemp, *School of Mines Quarterly*, ix., p. 74.

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sions imbedded in the limestone and in the beds of chert, the fragments of which are cemented together by a bluish gray, clayey mass. Cadmium-bearing blende occurs plentifully in two forms, a coarsely crystalline and a granular variety; pyrite is subordinate. To be noted is the absence of chalcopyrite and barite. Magnesite and calcite are found with the galena and often occur in crystalline masses in the dolomite. The following analyses by Spencer* show the character of the concentrates shipped to smelting works:

Name of Mine.	Henrietta.	St. Louis and Aurora	Coffman.	Stevenson and Wamper.	Magnet.	Ashcroft.
H_2O, hygr.	0.114	0.111	0.140	0.014	3.728	0.922
H_2O, comb.	0.407	0.706	0.188	.....	.....	.....
Insol.	1.050	0.912	1.201	.....	.....	.....
SiO_2	.....	.....	.....	0.908	1.257	0.208
S	13.948	14.537	12.364	14.040	14.351	13.722
As	.....	.....	.....	0.109	0.099	0.027
Zn	1.748	3.246	5.720	3.359	1.045	0.432
Pb	81.938	80.059	79.751	80.225	77.255	83.537
Fe	0.357	0.294	0.251	0.117	1.066	0.640
Al_2O_3	0.288	0.260	0.239	0.345	0.411	0.153
CaO	.....	.....	.....	0.396	0.228	0.003
CO_2	.....	.....	.....	0.316	0.184	0.002
	99.555	100.125	99.849	99.824	99.624	99.696

§ 21. Other Occurrences.—The lead deposits of Southwestern Texas in the Quitman Mountains are as yet of no special importance.

III.—Silver-Lead Ores of the Rocky Mountains.

To this division belong the occurrences of argentiferous lead ores in Colorado, South Dakota, Montana, New Mexico.

§ 22. Colorado.—The deposits of argentiferous lead ores of Colorado are more important than any others of the country.

Boulder County.—On the eastern slope of the Rockies the ores occur as veins in Archæan gneiss, which, on account of the indistinctness of the bedding, Emmons calls granite-gneiss. These are not real fissure veins but alterations of the country rock along certain planes. The mines are noted principally for the occurrence of rich telluride minerals, but in the Caribou district is found rich galena with silver minerals, such as stephanite and proustite. The Caribou mine contains a massive mixture of rich argentiferous galena, chalcopyrite, and blende, occurring in

* *Bulletin Missouri Mining Club*, I., No. 1, p. 30; No. 2, p. 51.

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gneiss near a dike of eruptive diabase. A shipment of ore made in 1891 contained: Pb, 44.1%; Zn, 2.5%; Fe, 9.1%; SiO_2, 8.6%; Ag, 72.0 oz. per ton.

The San Juan Region.—This embraces the southwestern part of Colorado, *i.e.*, the counties of Ouray, Hinsdale, San Miguel, Dolores, San Juan, and Montezuma. Characteristic of the region are immense quartz veins traversing old and young eruptive rocks. The productive ore bodies are found in the older massive and brecciated rocks, but veins occur also in the underlying gneiss and granite. In the neighborhood of Rico (Dolores County), sedimentary deposits occur in carboniferous limestone; in the Red Mountain district (Ouray County) deposits are found in cavities of silicified andesite. The deposits are mainly argentiferous. The minerals are argentiferous galena, silver-bearing gray copper, ruby silver, and native silver. Bismuth-silver minerals are frequent; small amounts of gold are found; blende occurs in considerable quantities. The gangue is quartz and kaolinite; barite is common; fluorite also occurs. In the bedded deposits of Rico iron and manganese are prominent. Near the outcrop the ores are often completely changed into a sandy carbonate.

Two important shippers from Creede (Hinsdale County) are the Amethyst and New York-Chance mines. The character of their ores is shown in the following table:

Name of Mine.	Pb.	SiO_2.	Fe.	Mn.	Zn.	S.	Ag.	Au.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Oz. per ton.	Oz. per ton.
Amethyst	6.0	68.1	4.6	6.0	.....	.....	71.7	0.055
New York-Chance	15.0	65.0	1.3	Trace.	2.6	4.1	60.0	0.175

The ores of the Amethyst mine often contain more lead than is shown in the analysis; sometimes they have practically none at all. The Ute & Ulay mine (Lake City, Hinsdale County) has furnished much desirable lead ore to smelters in the form of concentrates. They average 50 to 60% lead and contain 5 to 10% zinc. The following analysis* shows the general character of the ore: Pb, 61.35%; SiO, 12.2%; Fe, 2.4%; Zn, 6.7%; S, 10.6%; Ag, 21.05 oz.; Au, 0.08 oz.

* Private notes.

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Analyses from two characteristic carload samples of ore from the Red Mountain district by Kedzie* are subjoined.

	I.	II.		I.	II.
H_2O (by ignition)	6.75	11.94	SO_3	3.87	....
Insol	41.63	8.65	CO_2	2.39	....
Pb	18.40	....	Total	96.76	....
Fe_2O_3	17.12	64.80			
Al_2O_3	6.08	13.05	Ag, oz	128.00	68.00
MnO	0.59	1.98	Au, oz	0.22	0.10
CaO	1.70	....			
ZnO	1.23	....			

Custer County.—Two deposits of this region, the Bassick and Bull-Domingo mines, differ from other Colorado occurrences. The ore, which is found in large bodies without any definite boundary, forms concentric layers from $\frac{1}{8}$ to $\frac{1}{4}$ in. in thickness upon spherical fragments of eruptive country rock. The mines are located near Rosita and Silver Cliff respectively. At the Bassick mine the pieces of wall rock vary from $\frac{3}{8}$ to 24 in.; the sizes most common have diameters ranging from 4 to 12 in.; the rock proper shows only traces of precious metal. The metallic coating surrounds the pebbles and bowlders and fills the interstices between them. It forms concentric layers. The first one consists of sulphides of zinc, antimony, and lead, assays 60 oz. in silver and from 1 to 3 oz. in gold, and is dark in color. The next one is similar, but lighter in color, and assays higher; the third is blende, rich in silver and gold. Chalcopyrite with some pyrite often follows.

At the Bull-Domingo mine the occurrence is similar. Around a barren nucleus of hornblendic gneiss is deposited a coating of argentiferous galena, followed by siderite, but the ore contains no gold. The ore is dressed in a 150-ton mill, the concentrates produced assaying 60 to 65% lead and 15 oz. silver per ton.

The Terrible mine† at Ilse resembles the other two in some respects. A porphyry dike, 127 ft. wide, traverses the granite country rock for a considerable distance, 87 ft. being impregnated with crystals of cerussite. As mined the ore carries from 10 to 12% of lead and 1 oz. silver to the ton. It is concentrated to a product assaying 70% lead and 1.5 oz. silver.

Lake County.—Of the deposits in sedimentary rocks those of Leadville are the most important, producing about 60% of the argentiferous lead of Colorado. They occur principally at the

* "Transactions of American Institute of Mining Engineers," xvi., p. 581.

† Private communication of Messrs. Taylor & Brunton, Dec. 1, 1890.

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contact of the dolomitic "Blue Limestone" of the Lower Carboniferous, with an instrusive sheet of "White or Leadville porphyry" that covers it. They are, however, not confined to the contact, but extend into the limestone below. In some cases there is a gradual transition from ore to unaltered limestone; in others the ore forms pockets and caves in the limestone; again, the ore has entirely replaced the limestone and is included between two sheets of porphyry. It is exceptional when ore occurs unconnected with the contact surface in the form of irregular masses or running across the formation.

The principal ore is argentiferous galena with its secondary products of decomposition—anglesite, cerussite, and pyromorphite. Silver occurs in the altered ore principally as chloride and chloro-bromide, sometimes as sulphuret, rarely as chloro-iodide and native silver. It has been found that nodules of galena are richer in silver than cerussite. Small amounts of gold in the native state have been found in limestone, but it is usually associated with pyrite in porphyry. The accessory minerals are blende and its secondary products, carbonate and silicate of zinc. Arsenic is found as sulphide and in combination with iron as arseniate, antimony as sulphide, copper as carbonate and silicate, molybdenum and vanadium respectively as wulfenite and vanadate of lead and zinc and bismuth as sulphide and sulpho-carbonate. Iron and manganese occur as oxides. The gangue of the ore consists of silica (as chert, quartz, and combined with iron and manganese) and various clays charged with iron and manganese. Barite is not uncommon, but irregularly distributed. Siderite, pyrite, and gypsum are subordinate. A decomposed product, Chinese talc, consisting of a mixture of silicate and sulphate of alumina, occurs at the contact of the white porphyry and the limestone.

The following analysis of carbonate ore by Fluegger* represents an average sample of a thousand tons of ore coming from every producing mine. It shows the character of the ore mined at that time (1880).

	Per Cent.		Per Cent.		Per Cent.		Per Cent.
CO_2	5.58	FeO	0.89	MgO	3.04	Au	Trace
PbO	25.77	Fe_2O_3	24.86	As	0.01	Cu	Trace
Ag	0.31	MnO_2	4.03	Sb	0.02	Zn	Trace
SiO_2	22.59	Al_2O_3	3.99	$(KNa)_2$	0.98	H_2O	5.58
S	0.90	CaO	2.36	Cl	0.09	Total	101.00

* *Engineering and Mining Journal*, Jan. 8, 1881.

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The composition of the sample in its main features is therefore in round figures in percentage: Pb, 24; Ag, 90.5 oz.; (FeMn)O, 21.8; Al_2O_3, 3.4; (CaMg)O, 6.6; SiO_2, 22.6. Guyard,* comparing this analysis with a number of others of that time,† says the figures of lead, iron, and silica agree with the general composition of the smelting charges, but the silver is too high, the common relation being 1 oz. of silver to 6 lb. of lead, while Fluegger's analysis would correspond to 1 oz. to every 5 lb. The varying proportion of silver and lead in different parts of this district is shown by the following figures of Rolker:‡

Locality.	Mine.	Dry Weight. Tons.	Ounces Silver Per Ton.	Per Cent. of Lead.	Relation of Ag : Pb.
Fryer Hill.....	Chrysolite.....	10,561	65.45	21.45	1 : 6.5
Carbonate Hill.....	Evening Star.....	6,315	39.00	16.10	1 : 8
Carbonate Hill.....	Morning Star.....	4,794	25.00	38.45	1 : 32
Iron Hill.....	Iron Silver Mg. Co...	152,457	15.00	18.60	1 : 26

The table shows that Fryer Hill ores run higher in silver than either Carbonate Hill or Iron Hill ores, and that the latter two give a lower grade base bullion. The following analyses§ show the character of the carbonate ore of some of the leading producers in 1896:

Name of Mine.	SiO_2. Per Cent.	FeO. Per Cent.	MnO. Per Cent.	Pb. Per Cent.	Ag. Ounces Per Ton.	Au. Ounces Per Ton.
Northern (*a*).....	11.0	39.4	24.6	5.3	8.9	.....
Chrysolite (*a*).....	15.0	45.2	15.0	2.1	10.5	.....
Chrysolite (*a*).....	6.8	52.8	19.5	2.0	9.2	.....
Bison (*a*).....	9.0	29.0	31.0	0.6	10.2	.....
Bison (*a*).....	10.0	30.2	32.7	0.7	9.9	.....
Gray Eagle (*b*).....	12.2	24.4	0.4	18.5	30.3	.....
Maid of Erin (*b*)....	12.4	33.6 Fe	1.0 Mn	27.8	13.4	0.08
Starr (*b*).....	14.8	27.5 Fe	20.5 Mn	6.0	20.3	.....
Weldon II. (*b*).....	20.7	38.5 Fe	0.6 Mn	11.6	133.0	.....
Weldon III. (*b*).....	22.8	30.8 Fe	20.4 Mn	1.7	26.6	.....
Penrose (*b*).....	31.4	32.1	3.9	10.1	55.1	.....
Little Silver (*c*)....	67.6	21.6	3.1	0.0	24.6	.....
Ibex (*c*).....	68.3	15.8 Fe	0.2 Mn	4.0	25.8	1.90
Ibex (*c*).....	77.7	9.7 Fe	0.6 Mn	1.7	16.7	2.43
Black Prince (*c*)....	67.1	17.0 Fe	0.4 Mn	4.1	18.9	0.35

(*a*) Manganiferous iron ore. (*b*) Basic carbonate. (*c*) Silicious carbonate.

As the exploration extends deeper, the unaltered minerals,

* Emmons, "Geology and Mining Industry of Leadville, Colo.," Monograph xii., U. S. Geological Survey, Washington, 1886, p. 620.

† *Ibid.*, pp. 621-625.

‡ "Transactions of American Institute of Mining Engineers," xiv., p. 287.

§ Private notes.

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galena, blende, and pyrite are encountered. Unfortunately, at the same time, less silver is found. This transition from carbonate to sulphide ore is often very rapid. On Iron Hill, according to Blow,* the ore changes suddenly from a body of fine oxidized smelting ore to a close-grained sulphide ore consisting principally of zinc and iron sulphides with sulphide of lead in small quantities. The following analyses show the character of some of the sulphuret ores of Leadville:

	Galena.	Mixture.	Galena.	Mixture.	Lead Ore.	Zinc Ore.	Sulphide.	Sulphide Ore.
	Minnie Mine.	Minnie Mine.	Moyer Shaft.	Moyer Shaft.	Col. Sellers Mine.	Col. Sellers Mine.	Boreel Mine.	Average, 1896.
Pb....	72.65	50.86	44.0	15.0	27.40	10.70	11.8	12–18
Cu...	.....	.....	.....	.....	.....	.....	.....	1–3
Zn....	5.66	12.86	13.0	24.0	25.00	24.50	7.7	5–10
Fe....	1.60	9.30	11.0	16.0	6.00	16.60	23.8	77–59
S.....	15.66	24.50	30.0	40.0	35.00	40.00	.....	
Ag., oz	41.50	11.50	14.0	11.0	26.90	54.30	45.0	30
Au., oz	Trace.	Trace.	.....	.....	Trace.	Trace.	.....	.....
Insol..	4.12	1.88	.....	.....	3.00	3.40	14.7	5–10
H_2O ..	.....	.....	.....	.....	2.00	2.10	.....	.....
Ref....	(a)	(a)	(b)	(b)	(c)	(c)	(d)	(d)

(a) "Transactions of American Institute of Mining Engineers," xiv., p. 189, Freeland; (b) xiv. p. 288, Rolker; (c) xviii., p. 173, Blow; (d) Private notes.

Bartlett† gives as a fair example of Leadville zinky sulphide ore the following: ZnS, 28%; FeS_2, 36%; PbS, 16%; gangue, 20%, with 12 oz. silver and 0.05 oz. gold per ton. The lead concentrates from such an ore will contain PbS, 40 to 50%; ZnS, 15 to 20%; FeS_2, 30 to 50%, and 20 oz. silver and 0.1 oz. gold per ton.

The decrease in silver and lead and the increase in zinc with increasing depth have forced the question of concentration upon the mining companies. According to Ihlseng‡ several large and some small concentrating plants are enriching ores that run from 6 to 8 oz. in silver and from 6 to 10% in lead, saving 60% of the silver and 70% of the lead, 7 tons being concentrated to 1 ton.

Taylor and Brunton§ report that the Colonel Sellers mill, constructed by them, concentrates from 75 to 90 tons of sulphide

* "Transactions of American Institute of Mining Engineers," xviii., p. 170.

† "The Mineral Industry," v., p. 620.

‡ "Report of State School of Mines," Golden, Colo., 1887, p. 44.

§ *Engineering and Mining Journal*, May 8, 1886.

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ores, very free from gangue, and containing less than 10% lead and 9 or 10 oz. silver, into heads running 55 to 60% lead and tailings under 2% lead at a cost of 70 cents per ton. The following analyses by Keller show the character of the products obtained from coarse-grained ore.

	Crude Ore.	Blende.	Galena.	Pyrite.
SiO_2	0.960	1.090	0.120	0.190
Pb	16.185	3.525	79.958	9.592
Fe	22.951	10.864	2.303	39.431
Zn	19.246	47.522	1.734	3.043
Mn	1.664	1.477	0.154	0.406
Cu	Trace.			Trace.
As	Trace.			Trace.
S	39.240	37.040	15.760	48.196
Ag, oz.	7.5	3.0	19.8	5.2

Chaffee County.—The Monarch district with the Madonna, Eclipse, and Silent Friend mines was a very important producer of oxidized lead-silver ores which occurred in limestone. The mines have, however, seen their best days. The Madonna mine is the one best known. Its ore averaged 20 to 45% lead and 5 to 12 oz. silver per ton. An analysis by Sticht* gave the following composition: Fe_2O_3, 32.99%; Mn_2O_3, 0.93%; Al_2O_3, 2.99%; ZnO, 4.17%; CaO, 1.78%; $PbSO_4$, 12.47%; $PbCO_3$, 32.35%; SiO_2, 4.24%; SO_3, 3.30%; CO_2, 6.73%; H_2O, 6.68%; AgO, 0.23%(=6.8 oz. per ton).

Park County.—The deposits of this county have been developed in Archæan and Palæozoic rocks. The Mesozoic porphyries form dikes in Archæan and intrusive sheets in Palæozoic strata. Prominent are the deposits of Mount Lincoln and Mount Bross. The ores are argentiferous galena with sulphate and carbonate of lead and chloride of silver; pyrite, more or less decomposed, and associated with oxides of manganese, is also found, coloring the clay of the gangue; barite is of common occurrence. The deposits of Mount Lincoln form irregular bodies in the blue limestone, and are found generally near its upper surface; those of Mount Bross occur rather in the mass of the limestone near the dikes of porphyry. The character of the ore resembles, therefore, that of Leadville.

Pitkin County.—The Aspen deposits of silver-bearing ores came into great prominence in 1884. They occur in the same

* Dewey, "United States National Museum," Bulletin No. 42. p. 45.

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geological horizon as the Leadville deposits, viz., the Lower Carboniferous limestone, but they are not found in immediate contact with the eruptive rock. They occur principally between two varieties of limestone, the blue (a pure carbonate of lime) and the brown (dolomite), which is traversed by numerous small veins containing iron salts. The brown limestone breaks up by oxidation of the iron into small pieces, and is called "short lime." The ore is a limestone impregnated along certain fissures with fine-grained argentiferous galena. The main content of silver comes, however, from associated silver minerals, such as polybasite and stephanite. Near the outcrop the ore is somewhat decomposed, consisting of barite, carbonate of copper, oxides of iron and manganese, calcite, dice-shaped fragments of dolomite and galena.

The character of the ore is shown by the percentages of the following analyses furnished by Taylor and Brunton:*

$BaSO_4$...	0.5	5.5	9.1	11.7	13.0	15.3	24.2	26 0	26.2	29.5	30.0	26.9	40.0
SiO_2.....	6.5	15.4	9.2	20.0	19.7	27.9	17.3	40.0	44.9	14.3	7.0	23.3	19 0
CaO	7.6	27.5	29.8	19.0	15.7	14.2	13.7	3 0	1.8	7.8	11.1	12.9	12.5
Fe.....	9.0	5.4	5.5	4.0	6.3	6.5	1.5	9.4	10.6	2.5	7.3	4.8	6.5
Pb.....	19.3	.....	.....	.....	1.8	.....	4.7	8.0	.....	16.0	1.4	.....	.....
Ag, oz..	7.5	18.0	24.0	15.0	26.4	23.0	31.0	25.0	32.0	69.0	26.0	32.0	37.0
H_2O....	9.75	3.0	5.5	5.8	7.0	9.0	5.0	9.0	10.0	3.0	9.0	10.8	3.0

Eagle County.—The ores at Red Cliff on the Eagle River occur in Carboniferous limestone and on the contact between it and porphyry or quartzite. The ore is galena and anglesite. The production of the district has greatly fallen off; the Iron Mask mine is to-day the leading producer.

Summit County.—In the Ten-Mile district the ores occur in Upper Carboniferous limestone and in the superimposed sandstone. The typical mine is the Robinson. The ore is a rich argentiferous galena associated with pyrite and blende. It occurs near the surface of the limestone, in some cases extends into it, and in others penetrates it entirely.

§ 23. South Dakota.†—The occurrence of lead ores is confined to two small camps in the Black Hills, Galena and Carbonate. Argentiferous galena and carbonate ore occur in shoots in calcareous Potsdam rock, which overlies the upturned Archæan slates. These shoots are especially frequent where the porphyry

* Letter of Dec. 1, 1890.

† Carpenter, "Transactions of American Institute of Mining Engineers," xvii., p. 582.

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cuts through the rocks and where it overlies them. Pyrite and blende are found associated with the galena. The ore has to be closely sorted in order to make it assay 20 oz. silver and 50% lead.

§ 24. MONTANA.—The argentiferous lead ores of Montana are unimportant in comparison with the great copper deposits. They occur as metamorphic veins in crystalline rocks or as irregular ore bodies in limestones. The principal producing districts are Glendale (Beaverhead County), Wickes (Jefferson County), and Baker and Castle (Meagher County).

Glendale.—The ores occur interstratified in blue-gray limestone the age of which has not been determined. The minerals are argentiferous galena, blende, chalcopyrite, and pyrite, and their products of oxidation. The leading mines are those of the Hecla Consolidated Mining Co. First-class ore* shows SiO_2, 36%; Pb, 23.35%; Ag, 45.61 oz. per ton; low-grade ore with Pb, 7.2%, and Ag, 16.7 oz. per ton gives concentrates with SiO_2, 14.34%; Pb, 33.85%; Ag, 50.2 oz. per ton.

According to Lindgren† just south of Helena occur veins from 1 to 10 ft. in width, carrying galena, blende, chalcopyrite, pyrite, and arsenopyrite in a gangue of quartz. The galena assays up to 90 oz. in silver, the blende from 10 to 12 oz. in silver, the pyrite from 0.2 to 0.3 oz. in gold, the arsenopyrite 1 oz. in gold; the chalcopyrite carries more silver. The Helena Mining and Reduction Co. (Alta Montana mine) and the Gregory mines are the chief producers.

Castle.—The ores occur as chimneys in limestone and at the contact of limestone and igneous rock. They are oxidized to a considerable depth. The leading mines are the Cumberland and the Yellowstone.

The smelting works at Glendale, East Helena, and Great Falls obtain a large proportion of their ores from Idaho, especially from the Cœur d'Alene district.

§ 25. NEW MEXICO.—Although New Mexico produces much dry-silver ore, lead-silver ores are not of frequent occurrence. The principal ones are found in the Magdalena Mountains, (Socorro County), Lake Valley (Doña Aña County), and Cook's Peak (Grant County). The deposits near Magdalena occur in

* Ingalls, *Op. cit.*, p. 397.

† "U. S. Geological Survey: Mineral Resources of the United States," 1883-84, p. 422.

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limestone and porphyry and are from 4 to 40 ft. wide. The principal mines are the Kelly, the Juanita, and the Graphic. The ore averages 25% lead and runs very low in silver—8 oz. to the ton. Near Lake Valley occur deposits very similar to those of Leadville, Colo. They contain, however, very much less lead and are often free from it. The ores are principally oxides; silver is present as chloride and bromide, lead as cerussite and galena, the gangue is often rich in iron and manganese. The following partial analyses by Clark* show their general character:

Name of Mine.	SiO_2.	Fe.	Mn.	Pb.	Zn.	Ag. oz.
Sierra Grande.....	65	6	12	.....	.....	20
Sierra Bella.....	8 to 30	12	18 to 24	0 to 5	.....	20 to 50
Sierra Apache.....	28	14	18	.....	4	47

The district of Cook's Peak has become an important producer during the past few years. The Cook's Peak Mining Co. has made the largest output; its ore body lies in a fissure vein in porphyry; the mineral is argentiferous galena, the average grade of shipping ore from 1892 to 1894 being† Pb 43%, Ag 61.6 oz., Au 0.08 oz. per ton.

IV.—Silver-Lead Ores of the Pacific.

Argentiferous lead ores under this head occur in Nevada, Utah, Idaho, Arizona, and California.

§ 26. Nevada.—The production of argentiferous lead in Nevada has greatly diminished in the last 12 years. Although 31,063 tons of lead were produced in 1878, in 1890 the production had sunk to 1,994 tons. The ore comes principally from two mines, the Richmond mine and the Eureka Consolidated mine, of the Eureka district. It occurs in irregular chambers in the Prospect Mountain limestone of Ruby Hill, which is a compact magnesian limestone of the lower horizons of the Cambrian. In the upper part of the larger chambers it is in a loose state; in the lower ones it is more compact. It is an argentiferous and auriferous carbonate containing both anglesite and cerussite and a very small amount of galena; mimetite and wulfenite are often found. A remarkable feature is that at a depth

* "Transactions of American Institute of Mining Engineers," xxiv., p. 133.

† *Engineering and Mining Journal*, Jan. 6, 1894, p. 15.

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of 1,300 ft. the ore is still oxidized, and the regular sulphide from which it originated has not yet been reached. The associated minerals are pyrite, arsenopyrite, and blende, with their oxidized products. The gangue accompanying the lead ore is principally limonite, which colors it in different shades of yellow, red, and brown; lime, magnesia, alumina, and silica occur only in subordinate quantities. The character of the ore is shown by the following analysis made by F. Claudet* in 1878:

	Per Cent.		Per Cent.		Per Cent.		Per Cent.
PbO	35.65	Mn_2O_3	0.13	SiO_2	2.95	$H_2O + CO_2$	10.90
CuO	0.15	As_2O_5	6.34	Al_2O_3	0.64	Ag + Au	0.10
FeO	34.39	Sb	0.25	CaO	1.14		
ZnO	2.37	SO_3	4.18	MgO	0.41	Total	100.52

27.55 oz. silver per ton; 1.59 oz. gold per ton.

The average tenor of lead, silver, and gold is lower than the analysis shows, viz.: 15% lead, 23 oz. silver, 0.72 oz. gold.

The large amount of arsenic is to be noticed, which causes the formation of the unwelcome speise in smelting. Molybdenum has not been determined. Silver is present as chloride and sulphide; gold, probably finely divided, in the native state.

§ 27. UTAH.—The argentiferous lead deposits of Utah form more or less regular bodies in limestone or at the contact of limestone and eruptive rock. The ores are carbonate with anglesite strongly prevailing; the secondary minerals reach down to considerable depths, small amounts of galena occur. A very few of the mines have been worked down to the sulphuret ores. Others which formerly produced only silicious ores low in lead are running with greater depth into bodies richer in lead.

In Beaver County is the celebrated Hornsilver mine that closed down in 1885, but has again become one of the leading producers. The deposit forms a chimney between limestone and rhyolite. Anglesite and cerussite are the lead minerals; galena is found in small quantities; silver occurs as chloride, sulphide, and sulpharsenide; the gangue is calcite, quartz, barite and ferruginous clay. The subjoined analyses† by S. B. Newberry, made in 1879, show the character of the ore:

SiO_2.	$BaSO_4$.	$PbSO_4$.	Fe_2O_3.	Al_2O_3.	Sb_2S_3.	As_2S_3.	(CaMg)O	CO_2.	Ag.
Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Ounces.
15.17	0.49	74.51	4.80	1.71	0.37	1.12	0.50	0.62	78.33
47.95	2.71	28.80	.....	12.55	.....	.....	.....	.....	.....

* Curtis, "Silver-Lead Deposits of Eureka, Nevada," Monograph vii., U. S. Geological Survey, p. 60.

† Huntley, "Tenth Census of the United States, 1880," vol. xiii., p. 466.

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The average tenor of lead and silver in 1882 and 1883* was 37.80 and 36.83% lead and 34.2 and 27.15 oz. silver, which is considerably lower than the analysis of 1879. The present character (1897) of the ore shipped is shown by the following analyses:†

SiO_2.	Fe.	Zn.	S.	Cu.	$BaSO_4$.	Pb.	Ag.	Au.
Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.	Per Cent.	Ounces.	Ounces.
25.8	4.5	1.7	8.2	0.3	5.2	38.3	30.5	0.03
31.4	5.0	8.7	11.3	0.5	1.2	29.2	26.2	.....

The ores of the Tintic district (Juab County) are rather milling ores (Eureka Hill and Mammoth mines), but they are smelted at Salt Lake City with lead ores. Their composition is shown by the following analyses:

Name of Mine.	SiO_2.	Fe.	S.	Cu.	Pb.	Ag.	Au.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Ounces.	Ounces.
Eureka Hill.....	80.4	2.9	0.3	0.7	6.0	85.0	0.40
Eureka Hill.....	70.0	5.0	1.0	2.0	11.8	101.7	0.85
Mammoth.....	67.6	4.7	1.9	1.2	.....	35.8	1.24
Mammoth.....	67.4	3.3	2.6	12.2	2.0	32.2	0.56
Mammoth.....	86.6	4.5	.....	2.0	1.0	4.2	2.72

In Salt Lake County are the mines of Bingham and the Little and Big Cottonwood Cañons. The Old Telegraph mines of Upper Bingham Cañon are among the oldest of the State. The ore forms an irregularly mineralized zone between limestone and quartzite and sometimes porphyry. The limestone and porphyry are more or less decomposed near the deposits. The lead ore is carbonate containing more or less silica, ferruginous clay, and some galena. The silicious ore is an oxidized silicious pyrite, the quartz having become ochreous, spongy and brittle; the pyrite yellow, red or black ochre. With increase of depth the ores of the district have changed to sulphide and many of them must be concentrated before they can be shipped.

The following analysis made by Wuth‡ in 1876 shows the character of the ore:

	Per Cent.		Per Cent.		Per Cent.
$PbCO_3$	50.43	SiO_2	12.47	H_2O	0.19
PbS	15.02	Al_2O_3	3.01	Ag, oz.	21.14
Fe_2O_3	3.78	$CaCO_3$	3.64	Sb	Trace
CuS	0.67	$MgCO_3$	0.26	As	Trace
FeS	7.37	$CaSO_4$	3.04	CO	Trace

* "U. S. Geological Survey: Mineral Resources," 1883-84, p. 417.
† Private notes. ‡ Huntley, *Op. cit.*, p. 413.

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The first-class shipping ore in 1888 contained between 30 and 50% lead, 10 and 12 oz. silver, and 0.48 oz. gold per ton; the second-class ore is concentrated and runs a little higher in lead than the first-class. The ore of the Brooklyn mine, lying east of the Old Telegraph mine, is similar in character; first-class ore, 40 to 50% lead, 10 oz. silver per ton. Twenty tons of low-grade ore are concentrated into 4 to 5 tons of shipping ore. The following analyses* show the general character of the ores mined to-day (1897), in Bingham Cañon:

SiO_2.	Fe.	Zn.	S.	Cu.	Pb.	Ag.	Au.
Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Ounces.	Ounces.
18.0	22.5	3.0	n. d.	2.1	13.3	24.4	0.50
15.0	10.5	6.0	n. d.	1.2	40.5	35.4	0.10
7.0	40.2	.....	n. d.	1.0	2.8	7.5	0.10
17.2	17.0	7.3	n. d.	3.0	22.2	46.8	0.38
13.4	32.6	3.0	n. d.	1.2	6.7	10.0	0.26
8.0	34.2	2.0	n. d.	1.2	2.7	15.5	0.24
5.9	3.5	1.0	7.6	1.5	68.0	35.0	0.50
8.6	15.1	3.0	24.4	3.0	40.8	17.2	0.20
10.9	21.9	13.3	32.5	2.0	12.2	35.0	0.29

The first four analyses are from ores of the old Jordan and Galena mines; the first six analyses are from roasted ores.

In Summit County, near Park City, are the great veins of the Ontario and Daly Mining Co., carrying with free-milling ores argentiferous galena, which is separated by hand-sorting. Other mines have silicious lead ores which are enriched by wet concentration, the average shipping ore containing about 30% lead. The following three analyses of Ontario and Silver King ore give their general character:

Name of Mine.	SiO_2.	Fe.	Zn.	S.	Cu.	Pb.	Ag.	Au.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Ounces.	Ounces
Ontario.....	38.6	2.0	8.0	2.3	.....	30.6	70.8	0.10
Ontario.....	56.0	3.5	7.5	2.2	.....	15.0	74.0	0.10
Silver King.....	30.2	4.3	3.2	1.5	0.6	44.0	56.0	0.10

On account of the scarcity of lead in Colorado a good deal of Utah ore goes East to be smelted instead of being treated in and around Salt Lake City, which is the smelting district of the State. Formerly much ore from Idaho was shipped to Salt Lake City, but most of it now goes to Colorado and the Missouri Valley.

§ 28. IDAHO.—The argentiferous lead ores of Idaho come prin-

* Private notes.

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cipally from two districts, viz., the Wood River in the southeast and the Cœur d'Alene in the north.

In the Wood River district, limestone, shale, quartzite, and granite cut by eruptives are the principal rocks and rich argentiferous galena with its secondary minerals the principal lead ore. The silver-lead deposits, so far as observed by Eldridge,* are of two classes: those occupying fractured zones in calcareous shales, the metallic sulphides filling the interstitial places, and those occurring in beds of limestone which resemble very much the deposits of Leadville, Colo. Associated with the galena are blende, pyrite, arsenopyrite, tetrahedrite, erubescite and occasionally native silver. Ores run as high as 70% lead and 160 oz. silver per ton. Kirchhoff† says that the ore from the Hailey and Bellevue average 60% lead and from 80 to 100 oz. silver. Blake‡ writes that the Minnie Moore and the Queen of the Hills furnish concentrates assaying 65 and 62% lead, 80 and 65 oz. silver per ton. The leading towns are Ketchum, Hailey, and Bellevue; some ore is smelted near Ketchum, but most of it is shipped in the form of concentrates to smelting centers.

The Cœur d'Alene.—This district is situated in Shoshone County and the mines worked are in Wardner Cañon in the Bitter Root Mountains. The rocks are, according to Clayton,§ quartzite, magnesian shale, and schist, which have been much folded. The deposits form fissure veins varying in width from a few inches to 20 ft. The ore, a galena with its products of decomposition, is generally concentrated before shipment‖ to a product assaying from 60 to 65% lead, and from 25 to 30 oz. silver per ton.

The following analyses¶ give the general character of the concentrates shipped:

Name of Mine.	Pb.	Fe.	Mn.	Zn.	SiO_2.	S.	Ag.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent	Ounces
Tiger-Poorman	54.5	7.0	.....	10.0	4.2	.....	26.0
Bunker Hill & Sullivan.	62.0	10.1	1.7	.....	1.9	.....	29.1
Bunker Hill & Sullivan.	64.8	10.6	1.1	.....	1.8	16.9	30.0
Bunker Hill & Sullivan.	57.8	12.0	.....	.....	2.0	.....	30.0
Withheld	62.4	10.1	1.2	1.5	2.0	.....	21.5
Withheld	61.9	9.1	1.4	1.4	1.8	.....	21.7

* "U. S. Geological Survey: Sixteenth Annual Report," 1895, p. 264.

† "U. S. Geological Survey; Mineral Resources," 1887, p. 100.

‡ *Engineering and Mining Journal*, Nov. 29, 1890. § *Ibid.*, Nov. 23 and Dec. 14, 1890.

‖ "U. S. Geological Survey: Mineral Resources," 1887, p. 108; Clement, *Engineering and Mining Journal*, July 25, 1891. ¶ Private notes.

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§ 29. Arizona and California.—Very little lead is mined in Arizona or California. The smelting industry of Arizona is represented by the Benson and Tombstone works, but most of the ore goes to outside smelters. In the Tombstone district lead ores occur in a bluish-black limestone, probably of Carboniferous age, which has been much disturbed by faulting and is traversed by igneous rocks. Some deposits form vertical fissure veins, others follow the bedding planes.

The lead produced in California comes principally from the Selby Smelting Works at San Francisco, the principal one of the Pacific coast. In the southwestern part of the State, in Inyo and San Bernardino counties, occur the principal lead deposits; the ores of Inyo County occur either in limestone or in limestone associated with schist, those of San Bernardino County in dolomitic limestone.

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CHAPTER V.

RECEIVING,* SAMPLING, AND PURCHASING ORES, FLUXES, AND FUELS.

§ 30. Receiving and Weighing of Ores.—Ores arrive at the smelting works either in a loose condition or in sacks. The gross weight is taken on platform scales before unloading from the cars. The ore is marked, when unloaded, by a shingle 2 by 12 in., with the running lot number chalked on it. The entry is made in the receiving-book, a form of which is here given.

Ores contain a varying percentage of moisture; those arriving direct from the mine are often very wet or in winter frozen solid. After being weighed, the ores should be at once unloaded and the moisture sample taken. If the ore is in sacks, these should be emptied and weighed immediately, before they can become lighter by exposure to the air. They should be then dried and tied up in bundles. Generally speaking it is best not to receive an ore at all until the operations of weighing and unloading it, and taking out and weighing the moisture sample, can be performed in quick succession. If this is not possible, the weights will have to be taken after unloading, thus causing an extra handling.

§ 31. The Moisture Sample.—It is difficult to obtain an absolutely correct moisture sample, but a fair average may be arrived at which will satisfy the seller as well as the buyer. A correct moisture figure is necessary in making up the charges for the blast furnace, since the analytical data from which they are

* Austin, *Engineering and Mining Journal*, July 22, Aug. 5, 26, Sept. 16, 1882; Low, *Ibid.*, Sept. 24, 1881, Feb. 11, 1882; Reed, *School of Mines Quarterly*, iii., p. 253; vi. 351; Brunton, "Transactions of American Institute of Mining Engineers," xiii., p. 639; xxv., p. 896; Glenn, *Ibid.*, xx., p. 155; Hodges, *Engineering and Mining Journal*, Sept. 5, 1891; Bridgman, "Transactions of American Institute of Mining Engineers," xx., p. 416; Johnson *Engineering and Mining Journal*, Jan. 16, 23, 1892; Williams, *Colliery Engineer*, xvi., p. 1.

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figured refer to dry ore, and a correct allowance must be made for the moisture.

If the ore arrives loose, the surface which has been exposed to the air will be drier than the middle part, and this again less dry than the real average; if in sacks, it may show a greater irregularity, as more surfaces are exposed to the air. With ore of uniform size, arriving loose, the sample is obtained by taking a little from the shovel at regular intervals while the ore is being unloaded. Or, if it is discharged through a shoot into a bin below, it is better to take the sample at short intervals from the end of the shoot. When the same kind of ore arrives in sacks a small sample is taken from the top of each sack and a larger one from the center, or the sack-sample is taken from what seems to be an average part at once after emptying. The latter way may seem rather vague, but a person accustomed to emptying sacks will be able to take the correct sample. If the ore is not of uniform size it must be crushed in order to get a correct assay sample. In that case the moisture is best taken at the discharge of the crusher, cutting out the whole of the stream at short intervals of time in order to obtain a true average of coarser and finer parts. When the crushing cannot immediately follow the unloading, and the ore has to be temporarily stored in a bin to be sampled later on, the lumps may either be broken up and sampled with the fine part, or screened out and sampled separately, taking a correct proportion of coarse and fine in making up the final moisture sample.

In whatever way the sample may be taken, it should be put into a deep tin vessel with a closely fitting lid, so as to prevent the first part from drying while the other is being taken. The moisture sample must be taken at once to the assay office. A certain amount, commonly 50 oz., is then weighed out twice for two moisture determinations. These can be made when convenient, the weighed sample being transferred to a paper bag to await its turn. The scales commonly used are the Fairbanks, with sliding weight and beam divided to show the loss of weight in ounces and percentage.

Before the moisture is determined the weighed sample is reduced to a uniform small size. This is done on the grinding-plate, the ore being crushed there to the size of a pea. If it is too moist to be crushed it is first dried somewhat. The drying

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in small works is performed in a shallow tin or granite-ware pan on an oil-stove, a sand-bath, on top of the boiler, or by any means that permits the controlling of the temperature. Large works generally have a steam-heated drying closet. Such a closet should be 16 ft. long, 3 ft. wide, and 8 ft. high. At the top it should end in a hood from which a wooden box carries off the vapor. The closet has, say, four shelves, 14 in. wide, each consisting of ten 1-in. pipes. The temperature should be maintained close to 90° C. The samples are usually placed in iron trays on the shelves. It is necessary to stir the sample at intervals with a spatula or spoon so as to dry it completely. A cool beaker cover held a moment over the drying sample will show by the absence or presence of condensed vapor whether the drying is finished or not.

The following form of receiving-book for ores shows the entries that are necessary to give the requisite information:

Year.	Date of Receipt.	Shipper's		Company's Lot Number.	Gross Weight, Tons.	Tare, Lb.	Net Weight, Tons.	Moisture, Per Cent.	Dry Weight, Tons.	Placed		Remarks.
		Name.	Mark.							In Bin No.	On Bed No.	

§ 32. THE ASSAY SAMPLE.*—From a sample weighing a few ounces the metal value and the chemical composition of an ore are determined. The sample must therefore represent its average character. The most thorough way would be to crush the entire ore to a uniform small size, mix it, and then sample it down by one of the general methods to be discussed later on. If the ore is a sulphide, and has to be roasted before it is smelted, this preliminary fine crushing is necessary; if, however, as is frequently the case, the ore is treated raw, this crushing, which is wholly unnecessary, not only entails increased cost, but is disadvantageous, as fine ore disturbs the uniform running of the blast furnace, increases the amount of fluedust, and retards the smelting. The best ore for the blast furnace is between the size of a

* Austin, "Transactions of American Institute of Mining Engineers," xxvi., p. 388; Vezin, *Ibid*, p. 1095.

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man's fist and that of an egg; it will be desirable to reduce it as little as possible below this. When, however, the ore is rich and the metal-bearing part is irregularly distributed in the gangue, as is mostly the case, it is necessary to crush finer before reducing to a smaller quantity. In sampling, the sample will grow finer as the bulk is reduced, this occurring more rapidly with high-grade than with low-grade ore. In order to do accurate work it is necessary that a certain ratio between the weight of the sample and the size of the largest particle, once assumed, shall remain fixed through every stage of the operation, because the volumes and weights of pieces of the same kind of ore are to each other as the cubes of their diameters. Vezin, in 1866, finding that with pyritic ores of Gilpin County, Colo., running from 1 to 4 oz. of gold per ton, it was safe to cut down to 1 oz. a sample that had passed a 20-mesh screen, the diameter of the largest particle being 1 mm. ($\frac{1}{25}$ in.) prepared the following table for this class of ores:

Diameter of Piece, Mm.	1	2	4	8	16	32	64
Diameter of piece, inches	$\frac{1}{25}$	$\frac{1}{12}$	$\frac{1}{6}$	$\frac{1}{3}$	$\frac{2}{3}$	$1\frac{1}{3}$	$2\frac{2}{3}$
Minimum weight of sample, pounds	$\frac{1}{8}$	$\frac{1}{2}$	4	32	256	2,048	16,348

With richer ores, of course, it would be necessary to increase the weight of the sample.

Reed* arrived by calculation at certain definite maximum sizes which ores of a varying tenor in silver and gold may have when they are reduced from a 100-ton lot down to the assay sample. His tabulated results, slightly condensed, are subjoined:

Quantity of Ore. Reducing	Value of Silver in Ounces Per Ton.			Size of Ore.
	Highest: 300. Average: 50.	Highest: 3000. Average: 75.	Highest: 10,000. Average: 500.	
100 tons to 10 tons	Cocoanut	Fist	Fist	Maximum permissible size of ore for given grade.
10 tons to 1 ton	Orange	Egg	Walnut	
1 ton to 200 pounds	Walnut	Chestnut	Chestnut	
200 pounds to 5 pounds	Pea	Wheat	Wheat	
5 pounds to bottle-sample	20-mesh	25-mesh	50-mesh	

An ore must run very much lower than 50 oz. silver per ton, if the seller will be satisfied with cocoanut-size in reducing 100

* *School of Mines Quarterly*, vi., p. 357.

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tons to 10 tons; he will insist on at least fist-size, say 3 in. A 50-mesh sieve is generally considered to be too coarse for the final sample that is to be filled into bottles or envelopes; the 80-mesh and often the 100 and the 120-mesh sieves are in common use; the 80 to 100-mesh with ores free from metallics, and the 120-mesh with ores containing metallic silver and gold. Finally, Brunton* adds to the requirement of constant ratio between weight of sample and that of largest particle of ore contained in it, that the cutting down shall be regulated by the result obtained from dividing the grade of the richest mineral by the average grade of the ore, and also by the specific gravity of the richest mineral. He reasons that as the effect of a particle of the richest mineral is very much greater on the average of a low-grade ore than of a high-grade ore, this cannot be left out of consideration. With a high-grade the sample need not be crushed as fine as with a low-grade ore. The specific gravity of the richest mineral has to be taken into account, as the greater it is the greater will be its influence. From the consideration of these three points, Brunton derives a formula and then plots curves from which can be picked off the minimum weight a sample of ore may have before it needs re-crushing. The subjoined table, prepared by Mr. Fr. Drake for the writer, contains some of the leading figures of Brunton's curves for pyrite (sp. gr. 5.0), galena (sp. gr. 7.5), native silver (sp. gr. 10.5), and native gold (sp. gr. 17.5):

To explain the use of the table we may take the following example. A lead-silver ore, the average value of which is 40 oz. per ton, contains as richest mineral, galena assaying 2,000 oz. per ton. The $\frac{\text{grade of richest mineral}}{\text{average grade}} = \frac{2000}{40} = 50$ and the specific gravity of galena=7.5.

In order take an 800-lb. sample, the ore will have to be crushed down to 0.5 in. which corresponds to 793 lb. It will be noticed that Brunton's table requires much larger quantities of ore in order to obtain a satisfactory sample than either Vezin's or Reed's.

A 100-ton lot of medium-rich ore, instead of being sampled as a whole, is often divided into two lots and sampled separately.

* "Transactions of American Institute of Mining Engineers," xxv., p. 826.

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Specific Gravity Richest Mineral	Size of Particles. Mesh.	Size of Particles. Diameter. Inches.	Safe Weight in Pounds for Sample whose Largest Particles are of Size given in Second Column. Grade of Richest Mineral Divided by Average Grade. 10	50	100	400	700	1000	1500	2000	2500
5.0	120	.0043				.0051	.0089	.0123	.0198	.0257	.0334
	100	.0055	.00023	.00127	.0025	.0108	.0181	.0259	.0389	.0518	.0674
	20	.027	.0248	.1352	.273	1.1	1.9	2.7	4.1	5.5	7.1
	8	.062	.279	1.5	3.	12.	21.	31.	46.	62.	80.
	4	.145	2.81	15.	30.	124.	218.	312.	468.	624.	812.
	2	.338	30.	163.	330.	1331.	2331.	3332.	5000.	6668.	8670.
		.5	78.	429.	868.	3501.	6133.	8766.	13153.	17541.	22806.
		1.	486.	2646.	5346.	21546.	37746.	53946.	80946.	107946.	140346.
		2.	3888.	21168.	42768.	172368.	301968.	431568.	647568.	863568.	1122768.
		4.	31104.	169344.	342144.	1378944.	2415744.	3452544.	5180544.	6908544.	8982144.
7.5	120	.0043				.01156	.02026	.0289	.0434	.0579	.0724
	100	.0055	.00054	.00295	.00597	.0241	.0421	.0602	.0904	.1205	.1507
	20	.027	.0573	.3116	.6296	2.5	4.4	6.3	9.5	12.7	15.
	8	.062	.621	3.3	6.8	27.	48.	68.	103.	137.	172.
	4	.145	6.2	33.	68.	276.	488.	691.	1037.	1383.	1729.
	2	.338	61.	337.	681.	2745.	4810.	6875.	10316.	13757.	17198.
		.5	145.	793.	1608.	6463.	11333.	16188.	24288.	32388.	40483.
		1.	729.	3969.	8019.	32319.	56619.	80919.	121419.	161919.	202419.
		2.	5832.	31752.	64152.	258552.	452952.	647352.	971352.	1295352.	1619352.
		4.	46611.	253770.	512721.	2066421.	3620121.	5173821.	7763821.	10352821.	12942321.
10.5	120	.0043		.00287	.0058	.0234	.04096	.0585	.0878	.1171	.1464
	100	.0055	.00109	.00599	.0121	.0487	.0854	.1221	.1833	.2443	.3054
	20	.027	.1108	.6007	1.21	4.8	8.5	12.	18.	24.	30.
	8	.062	1.17	6.3	12.8	51.	91.	130.	195.	260.	325.
	4	.145	11.4	62.	126.	509.	891.	1274.	1912.	2550.	3188.
	2	.338	110.	600.	1213.	4891.	8569.	12247.	18377.	24507.	30637.
		.5	242.	1319.	2665.	10742.	18819.	26896.	40357.	53819.	67289.
		1.	1020.	5556.	11226.	45246.	79266.	113286.	169986.	226686.	283386.
			2000	10000	15000	20000	25000	30000	40000	50000	60000
17.6	120	.0043	.181	.9050	1.8	1.8	2.2	2.7	3.6	4.5	5.4
	100	.0055	.37	1.88	2.8	3.7	4.7	5.6	7.5	9.4	11.3
	20	.027	37.	189.	285.	380.	475.	570.	760.	950.	1140.
	8	.062	397.	1989.	2985.	3980.	4975.	5970.	7960.	9950.	11940.
	4	.145	3818.	19098.	28650.	38200.	47750.	57300.	76400.	95500.	114600.
	2	.338	35292.	176082.	264150.	352200.	440250.	528300.	704400.	880500.	1056600.

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Uniform low-grade ores are, however, frequently sampled in 200 and 250-ton lots. With very rich ores, or with ores difficult to sample, as, for instance, Cripple Creek gold ores, the entire lot is often taken as a sample. Before sampling carbonate ores that are dry, they should be moistened with a hose, so that the fine dust, which often represents the richest part of the ore, may adhere to the lumps. If this is not done, it will be partly lost by being blown away when the ore is handled, or it may drop through the ore-heap to the floor and be partly lost there. In any case, the dust is likely to be unevenly distributed, and thus errors may occur. For instance, a lot of ore that sampled wet gave 400 oz. silver to the ton, sampled dry only gave 390 oz. The same holds good in sampling telluride gold ores where the dust is likely to be very rich in comparison with the other parts of the ore.

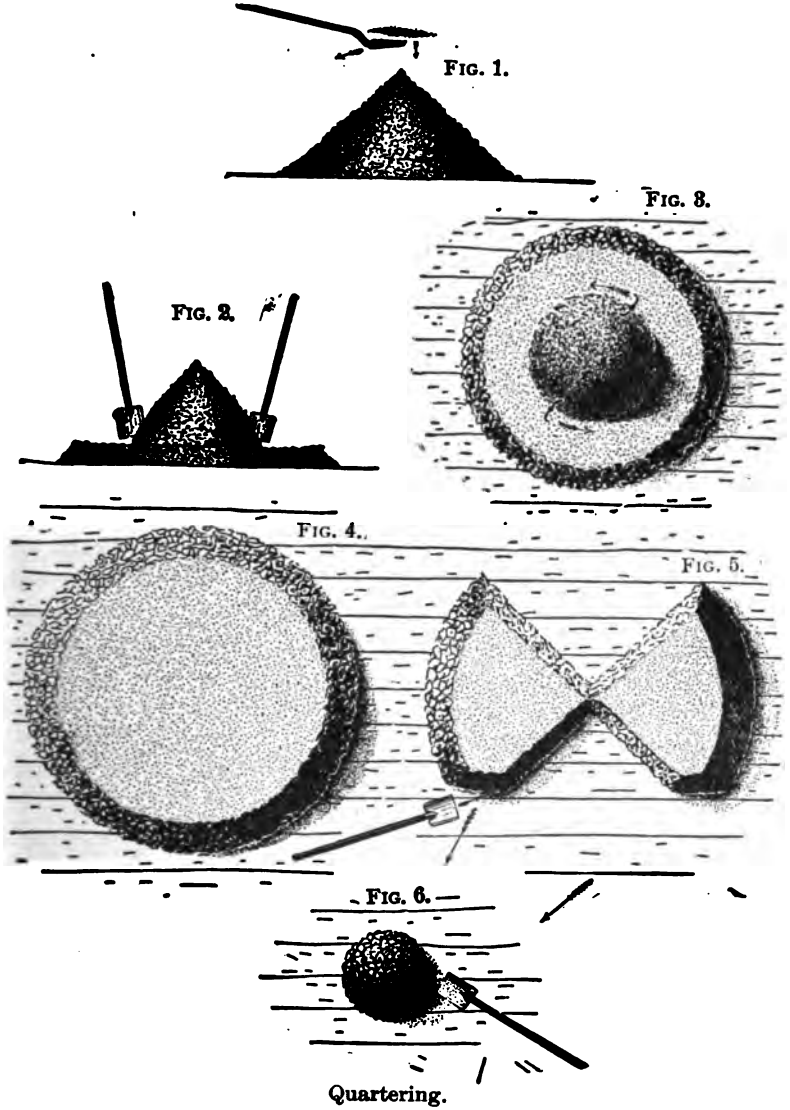
The breaking of the ore down to walnut-size is commonly done in a crusher; from this down to the size of wheat between rolls; smaller sizes are obtained by grinding in a mill or on a plate.

Sampling is done either by hand or machine. There are five ways of doing it by hand: by quartering, by fractional selection, with the split shovel, by channelling, and with the riffle. Machine-sampling is of two kinds: that which divides the stream of ore into unequal parts and takes off continuously the smaller one as the sample, and that which takes intermittently at certain intervals the whole of the stream as sample.

Quartering.—By this method the ore is reduced one-half in quantity at a time. It presupposes a thorough mixing of the ore, which is done by coning. The ore coming from the shoot of a crusher or of rolls is wheeled away and dumped in a circle large enough for the two samplers to be able to stand inside. They work as partners, walking around the circle while forming the cone, remaining always diametrically opposite each other while they pile up the ore. In doing this, care must be taken that every shovelful is thrown directly upon the top of the forming cone (Fig. 1), so that it may run and spread evenly down the sides. With experienced men the circumference of this cone will be a true circle. When the ore is exhausted the floor is swept carefully and the sweepings deposited on the top of the cone—not swept simply toward it—forming a ring of fine ore. A 50-ton cone is from 10 to 12 ft. high. The shovels used are long-

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handled and round-pointed. The ore being thus mixed, the cone is pulled down, the men working in opposite pairs (Figs. 2



Quartering.
FIGS. 1 TO 6.

and 3). They begin near the top of the cone, and, walking round it, work it down from center to periphery until it becomes

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gradually transformed into a truncated cone from 6 to 18 ft. in diameter and from 6 to 12 in. in height (Fig. 4). A short-handled, square-pointed shovel is desirable for this work. With the sharpened edge of a straight lath two diametrical lines are drawn across the cone at right angles to each other. The ore is thus divided into four equal parts (hence the name quartering). Two opposite sections (Fig. 5) are removed to the bin, while from the other two a new cone is formed (Fig. 6), and the process of quartering repeated. When the pile has become so small in quantity that it has to be crushed to attain a uniform sample, this is done by crusher or by rolls. The whole process is repeated until the sample is reduced to 100 or 50 or less pounds, forming the finishing sample, which is treated separately. Care should be taken to avoid losing any dusty part of the ore or working other material into the sample, the danger of error increasing as the sample becomes smaller. Two men will quarter a 1-ton sample down to a few pounds in about two hours.

Quartering was formerly used as a starting method for the entire mass of the ore as it came to the smelter; it has generally become a finishing method for reducing a part only of the ore. The method is satisfactory if carried out with the necessary care. There is, however, danger that the point of the cone may shift away from the center, thus making the quarters of uneven size. A rod can be driven into the floor to serve as a guide in coning, but this is rarely done; and even if the rod is used the men are very apt to shovel so that the ore is unevenly distributed. Another difficulty is that with ore of medium size, when the cone has been pulled down and the two quarters are being taken out, it is hard to prevent some of the coarser parts of the remaining quarters from rolling down into them. (This can be obviated with small lots by dividing the truncated cone with sheet iron plates or wooden laths to keep the quarters separate.) Lastly, the method is slow and requires much space. These objections have brought it into less general use than a few years ago.

Brunton* has patented† a quartering shovel by whose use the operation is very much shortened. It is (Fig. 7) a flat-bottomed steel shovel, 10 in. wide, turned up at the sides and divided into three compartments. The center compartment is closed at the

* *Engineering and Mining Journal*, June 20, 1891.
† Patent No. 454,120, June 16, 1891.

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back and has the width of one-quarter of the shovel. Thus, when the shovel has been pushed into a heap of finely crushed ore and filled, it is raised, and a "sharp rotary motion to the right" will discharge the ore from the outside compartments to one side, forming the rejected ore pile, and leave the sample in the central compartment to be emptied on the other side. In a speed test a sample of one ton of ore was cut down with this shovel to 100 lb. in 15 minutes by one man.*

Fractional Selection.—This is a starting method. It consists in taking, while unloading, every second, third, fourth, fifth or tenth shovelful of ore for the first sample, the number of shovelfuls taken depending on the richness of the ore and the more or less even distribution of precious metal. The sample is then cut down in the same way. If the shoveling is done every time from



FIG. 7.—BRUNTON'S QUARTERING SHOVEL.

the floor, as it should be, the ore without previous mixing will give a correct sample. In comparison with quartering the method gives more leeway, as from 50% down to 10% of the ore can be taken as a sample; it is from three to four times as quick, requires little floor space, and little or no skill from the workman.

The same method is used with sacks in making the first sample if the ore is known to be uniform in character, as otherwise the chances of error are too great.

As in quartering when the sample becomes small it has to be crushed. Fractional selection is rarely, if ever, used for finishing a sample.

Split Shovel.—The tool (see Figs. 8 and 9)† used resembles a fork with a long handle, the prongs being replaced by from four to six troughs made of sheet iron. The width of the troughs,

* The writer is informed by disinterested parties that the results with the Brunton shovel check well with their own results, obtained from regular quartering.

† *School of Mines Quarterly*, III., p. 267.

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which is equal to the distance between them, depends on the size of the ore to be treated, and the largest piece of ore in the heap ought not to exceed one-fourth the width of the trough.

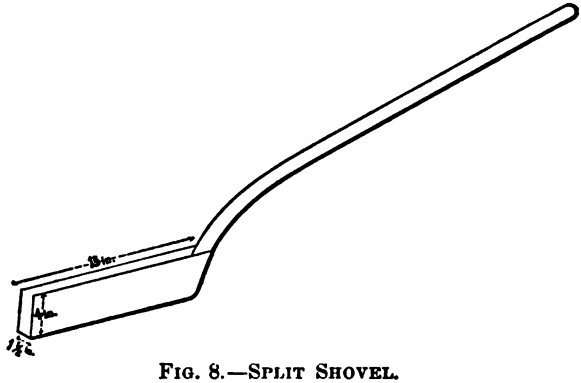


FIG. 8.—SPLIT SHOVEL.

Thus if the ore is ½ in., a 2-in. trough is required. The depth of a trough is such that a piece of ore striking the bottom will not rebound and fly out—say from 2 to 4 in. deep for a 2-in.

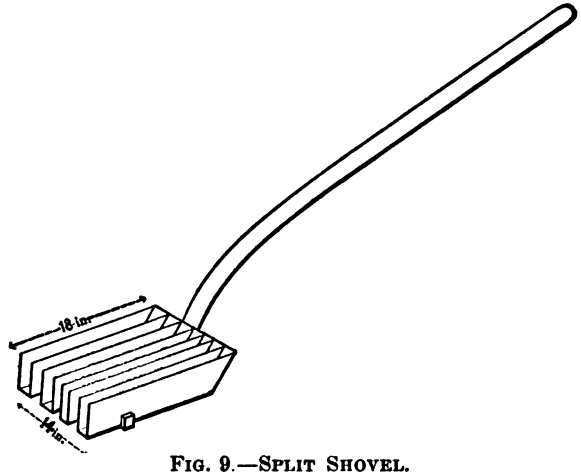


FIG. 9.—SPLIT SHOVEL.

trough. The length of the trough varies from 15 to 18 in; the handle is that of a long-handled shovel. In sampling, the split shovel is placed on the floor by one man, while the other, the sampler, facing the ends of the troughs, delivers the ore from a

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square-pointed shovel in a thin stream in the direction of the troughs. One-half of the ore is caught in the troughs, one-half passes into the spaces between. Re-sampling over the fork is continued until the troughs are full, when it is lifted out and the contents discharged on a separate heap, forming the reduced sample. The whole heap is passed over the fork in this manner, and the first reduced sample cut down similarly until it is necessary to crush finer, when it is passed over another split shovel with smaller-sized troughs—say from ¾ to 1 in. wide.

It is evident that sampling with the split shovel can be used as a starting method only with pretty fine material, and this must be very well mixed, if the sample is to be accurate. Smelters make the objection to the split shovel that an undue amount of fine ore is liable to be caught by the troughs. This gives an incorrect sample, as the fine ore assays differently from the coarse ore and is usually richer. The split shovel is used at some sampling works for the entire sampling, but this is not common. It is rather the tool for finishing a sample, and is even used in the laboratory in a modified form as assayer's riffle.

This method works as fast as quartering, but more slowly than fractional selection. If the split shovel takes out one-fifth of the ore as a sample, it amounts to the same thing as reserving every fifth shovel in fractional selection; in the former case one man does the shoveling while the other holds the split, in the latter both men shovel and each works more quickly than is possible with the split shovel.

Channelling.—This consists in spreading out the crushed and thoroughly mixed ore in a square, about 4 in. thick, and then taking the sample out in parallel grooves—say 1 ft. apart—across the square, first one way, then the other. The sample is reduced by repeating the same operations. The greater the number of the grooves (channels) drawn, the less thorough need be the preliminary mixing of the ore.

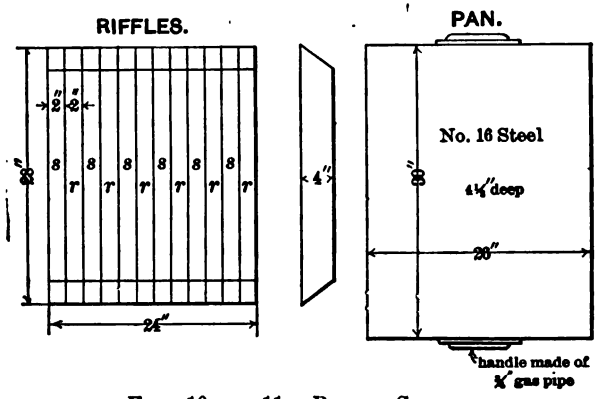
This method, used formerly at Batopilas and Silver Islet,* is used in hardly any silver-lead smelting works now as a starting method, as it requires much floor space, and is slow and inaccurate on account of the difficulty of preventing coarse pieces from falling into the grooves made in the operation. The method is still in use as a finishing method, where the mixing of a small

* Lowe, *Engineering and Mining Journal*, Sept. 24, 1881.

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amount of fine ore is easily effected, but even here it has to yield to other methods.

Riffle Sampler.—This simple apparatus (see Figs. 10 and 11) with the receiving pan consists of a series of trays or riffles, made of No. 16 steel, alternating with the same number of open spaces. The finely crushed ore is fed upon the riffle, when



FIGS. 10 AND 11.—RIFFLE SAMPLER.

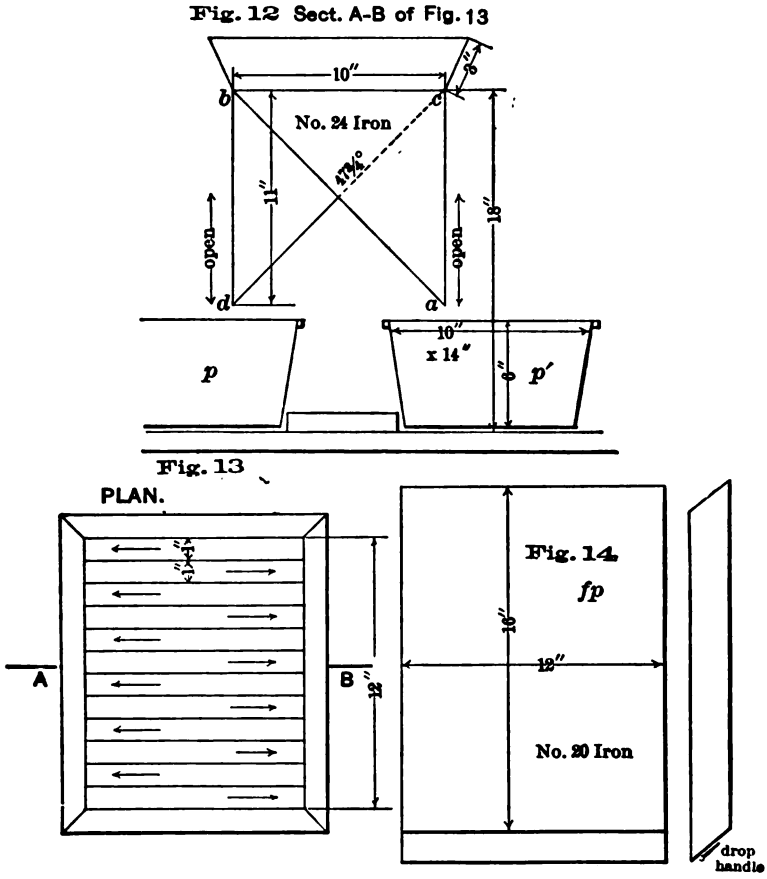
one-half will drop through the open spaces and will be collected on the receiving pan. Riffles and spaces have been made ½ in. wide, which, however, is rather small, a common size being 1 in. The riffle sampler is only used for finishing.

An improved form of riffle is the one going by the name of Jones' Sampler (Figs. 12, 13 and 14). It may be considered as consisting of a number of inclined riffles pointing alternately in opposite directions, the whole resting on a frame. In Fig. 21 the triangle *a b c* forms the first riffle, discharging to the right, triangle *b c d* the second, discharging to the left, and so on. The plan (Fig. 13) shows the sampler to consist of 12 riffles, six of which discharge to the right and the rest to the left. The ore to be halved is placed on the charging pan (Fig. 14) and fed upon the sampler, when the rejected ore will be collected in the receiver *p* and the sample in *p'*. In order to do the best work, four receivers forming a nest are desirable. From the size of the apparatus it is clear that it is to be used only for finishing a sample. It is, however, constructed in larger proportions and is used for cutting down samples that are too small

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to be worked by mechanical sampling. The apparatus is simple, permits quick and accurate work, is easily handled and cleaned, and is very satisfactory as long as the ore is crushed sufficiently fine.

Continuous Mechanical Sampling.—A number of machines*



FIGS. 12, 13 AND 14.—JONES' SAMPLER.

for continuous sampling have been invented, but few are in use to-day. The drawback to the method is that a falling stream of ore is never uniformly mixed. If it glides down in an inclined spout, the fine particles will either remain on the bottom, while

* Egleston, *Engineering*, Dec. 15, 1876; Reed, *School of Mines Quarterly*, iii., p. 253.

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those that are coarse will rise to the top, and often bound along over the surface, or the fine, slow-moving particles will be pushed to the side by the coarse ore passing rapidly down the center. In order to counterbalance this disadvantage, a preliminary fine crushing and sizing becomes absolutely necessary to obtain a sample that is approximately satisfactory. Why fine ore is

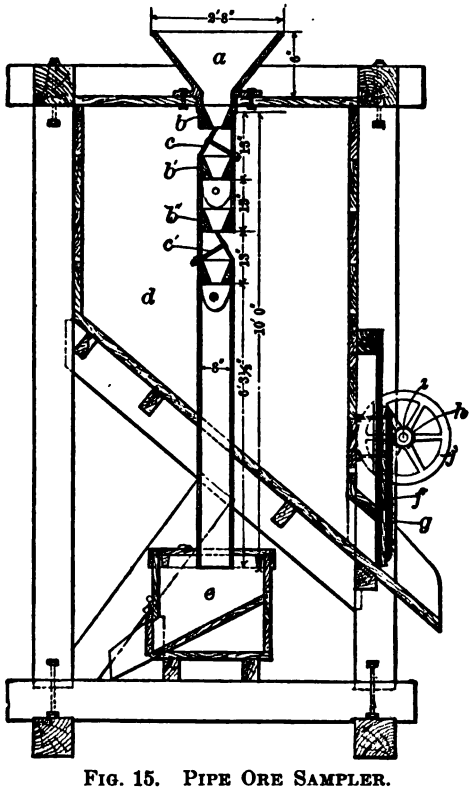


FIG. 15. PIPE ORE SAMPLER.

undesirable for the blast furnace has already been discussed. Continuous mechanical sampling may perhaps be advantageously used for gold-mill concentrates and finely crushed matte, but not for ore.

The pipe ore sampler, as represented in Fig. 15,* is an apparatus of this class which was much used in lead-silver works before the many disadvantages of the method were fully recognized.

* Taken from a drawing of Messrs. Fraser & Chalmers, Chicago, Ill.

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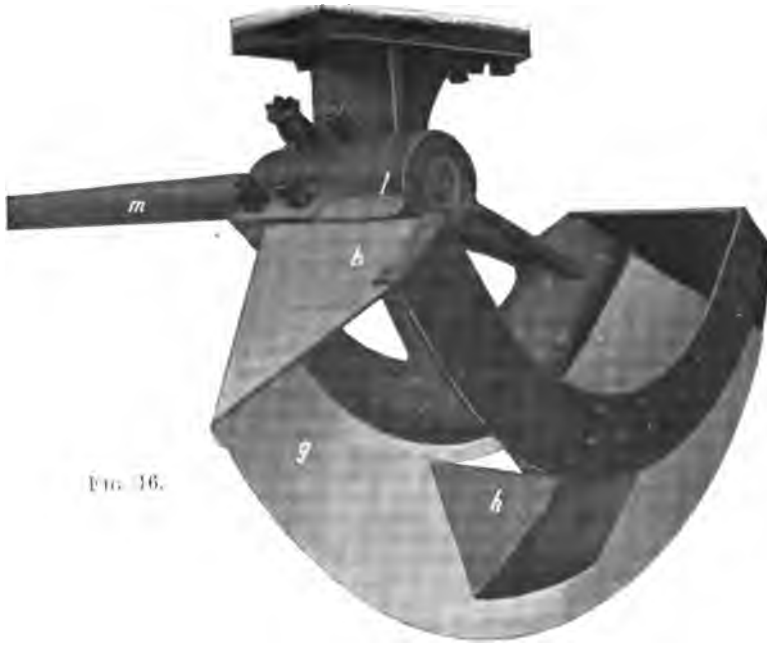
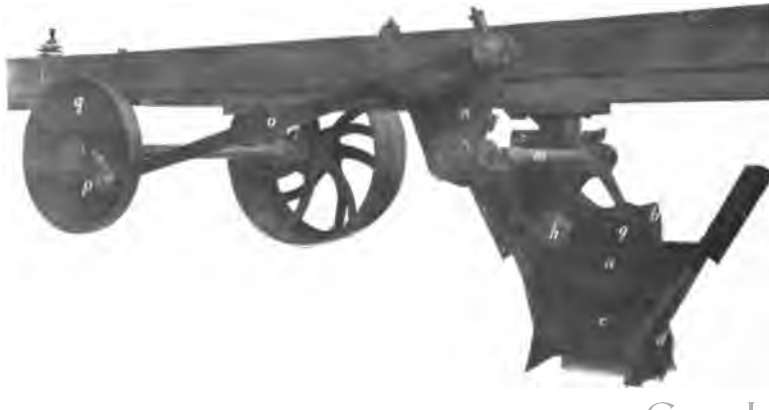


Fig. 16.



Fig. 17.



Figs. 16 to 18.—The Brunton New Mechanical Sampler.

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The finely crushed and sized ore is fed into the hopper *a*, ending in a small funnel *b*. Through this it passes over the divider *c*, which cuts it into two equal parts. One-half is discharged into the ore-bin *d*, while the other descends in the pipe and is thoroughly mixed by passing through the funnel *b'*, and then being scattered toward the sides of the pipe before it is brought together again in a small stream by the funnel *b''*, which delivers it to the second divider *c'*. The rejected ore falls into the bin *d*, while the sample is collected in the sample-bin *e*, to be put again through the sampler until it has been sufficiently reduced in size. The ore-bin is closed by a wooden gate *f*, running in heavy iron guides. This is raised and lowered with rack and pinion (*g*, *h*), the pinion-shaft *i* being turned by a hand-wheel *j* that is keyed to it. The sample-bin *e* can be locked, if desired.

Intermittent Mechanical Sampling.—This method gives better results, as the entire falling stream of ore is deflected at certain intervals to cut out the sample. When crushed ore is flowing continuously, it changes from rich to poor and gradually back again. In taking out equal portions all across the stream the slices taken will vary but little in value from those adjoining, and if taken at regular intervals the sample obtained will be a true average. The amount of ore taken out for the sample is regulated by the number of deflections that take place and the length of time they last. The advantage is that the bad effect of the irregular distribution in the stream of ore is neutralized, and thus fine crushing and screening made unnecessary. Rittinger* describes an apparatus of this kind. Four modern machines by Brunton, Bridgman, Vezin, and Constant may serve as examples.

Brunton's New Mechanical Sampler.—Brunton has constructed two mechanical samplers. In the older machine† the ore coming down through a spout was divided into two parts by a reciprocating tongue which deflected the entire stream of ore alternately to right and left. The edge of the tongue in traveling across the stream from right to left and returning made two opposite diagonals. The sample section, therefore, had the form of a rhomboid instead of that of a rhombic prism, and could not represent an accurate sample. In the modern machine,‡

* "Lehrbuch der Aufbereitungskunde," Berlin, 1867, p. 583.

† "Transactions of American Institute of Mining Engineers," xiii., p. 639; this book, first edition, p. 55. ‡ U. S. Patent No. 553,506, Jan. 28, 1896.

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represented in Figs. 16 to 18, this is not the case. It consists of a sheet-iron box *a*, in which a deflector *b* rocks to and fro in the path of a descending stream and takes out the sample. The box has slanting sides and a bottom of the form of an inverted V, the limbs of which end in shoots receiving the sample *c* and the rejected ore *d*. The stream of ore, delivered at the top of the box into a shoot *f*, meets the deflector, the essential part of the machine. It consists of a back plate *g*, inclined to one side, and a pocket *h* attached to it, slanting in the opposite direction. Thus three divisions are formed, two *i* and *j* on the outside for the rejected ore and a central one *k* for the sample. The back plate is attached by means of the hub *l* to the shaft *m*, receiving through crank *n*, pitman *o*, wrist *p*, and driving wheel *q*, a reciprocating motion. As the deflector rocks to and fro under the descending stream of ore, the rejected ore cut out is delivered into the shoot at one side, the sample into that at the other. The sample-pocket, in passing underneath the stream of ore, strikes it first with one, then with the other of its parallel sides, and cuts it equally in all parts in the form of a rhombic prism. The ratio of rejected ore to sample is governed by the width of the sample pocket and by the length of the arc through which the deflector travels. A common ratio is 5 : 1. The machine is giving satisfaction with Cripple Creek gold ores, probably the severest test it can be put to.

Figs. 19 and 20 represent two vertical sections through the Taylor and Brunton Sampling Works of Victor, Colo., in which the new Brunton mechanical sampler has been introduced. The works have a capacity of 250 tons in 10 hours and require a 35 horse-power engine. The cost of a plant of this size in Colorado is about $40,000. In a 10-hour shaft one man attends to the mechanical part of the sampling, who also takes moistures and finishes the sample. About 20% of the ore is cut as first sample, thus delivering to the smelter about 80% of a shipment in coarse sizes as discharged from the crusher. In the further reduction of the first sample the proportion of rejected ore and sample remains the same, and the ore is re-crushed before every cutting down. The sampling is entirely automatic, the ore once fed into the crusher not being handled again, and the sample, reduced to $\frac{1}{625}$ of the original weight of the ore, is ready to be

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ground fine and further reduced in volume by one of the finishing methods.

The ore, as it arrives in cars or wagons, is weighed, unloaded and fed into a 9 by 15 in. Blake crusher *A*, set to 1½ in., the pulley making 260 revolutions per minute. The crushed ore falls into the boot of the belt elevator *B*, which runs at a rate of 333 ft. per minute and is delivered to the sampler S_1. The rejected portion passes through the shoot *h* into hopper *H*, while the sample is discharged through shoot *m* to the coarse rolls R_1, 14 by 36 in., and set to 1½ in., the pulley making 75 revolutions per minute. The rolled ore goes to the belt elevator B_{11} running with a speed of 333 ft. per minute. This delivers the first sample to sampler S_{11}, from which the rejected ore drops through shoot *n* into hopper *H*, while the second sample passes on to the oscillating feeder *f* which discharges to the medium rolls R_{11}, 14 by 27 in., set to ¼ in., the pulley making 100 revolutions per minute. The second sample now goes to sampler S_{111}, delivering the rejected ore through shoot *o* into hopper *H*, while the third sample passes through the steam-heated trough *T* with screw-conveyer in which it is thoroughly mixed and dried, and then fed to the fine rolls R_{111}, 10 by 20 in., set to $\frac{1}{16}$ in. and making 125 revolutions per minute. From the rolls it passes into sampler IV, whence the rejected ore and the fourth sample finally drop into their respective receivers *D* and *E*.

*The Bridgman Sampler.**—Type *A*, represented by Figs. 21, 22 and 23, approximately drawn to scale, may serve as an example of the machine. It occupies a floor space 3 by 4 ft., is 7 ft. 6 in. high, and costs $1,000. It consists of three hollow truncated cones, I., II., III. (called apportioners), driven by the pulley *F*, and three stationary concentric receptacles, R_1, R_2 and *H*. The first two discharge the original and duplicate samples they receive through the spouts T_1, T_2, into the sample-boxes Z_1, Z_2; the third discharges the rejected ore that falls into it through the shoot *S*.

Apportioners I. and III. (Fig. 23) move in the same direction, making respectively 5 and 45 revolutions per minute; apportioner II., moving in the opposite direction, makes 15 revolutions. Apportioner I. (Fig. 23) consists of two concentrical rings having eight compartments, L_1 to L_8. To each of these an adjusta-

* "Transactions of American Institute of Mining Engineers," xx., p. 416.

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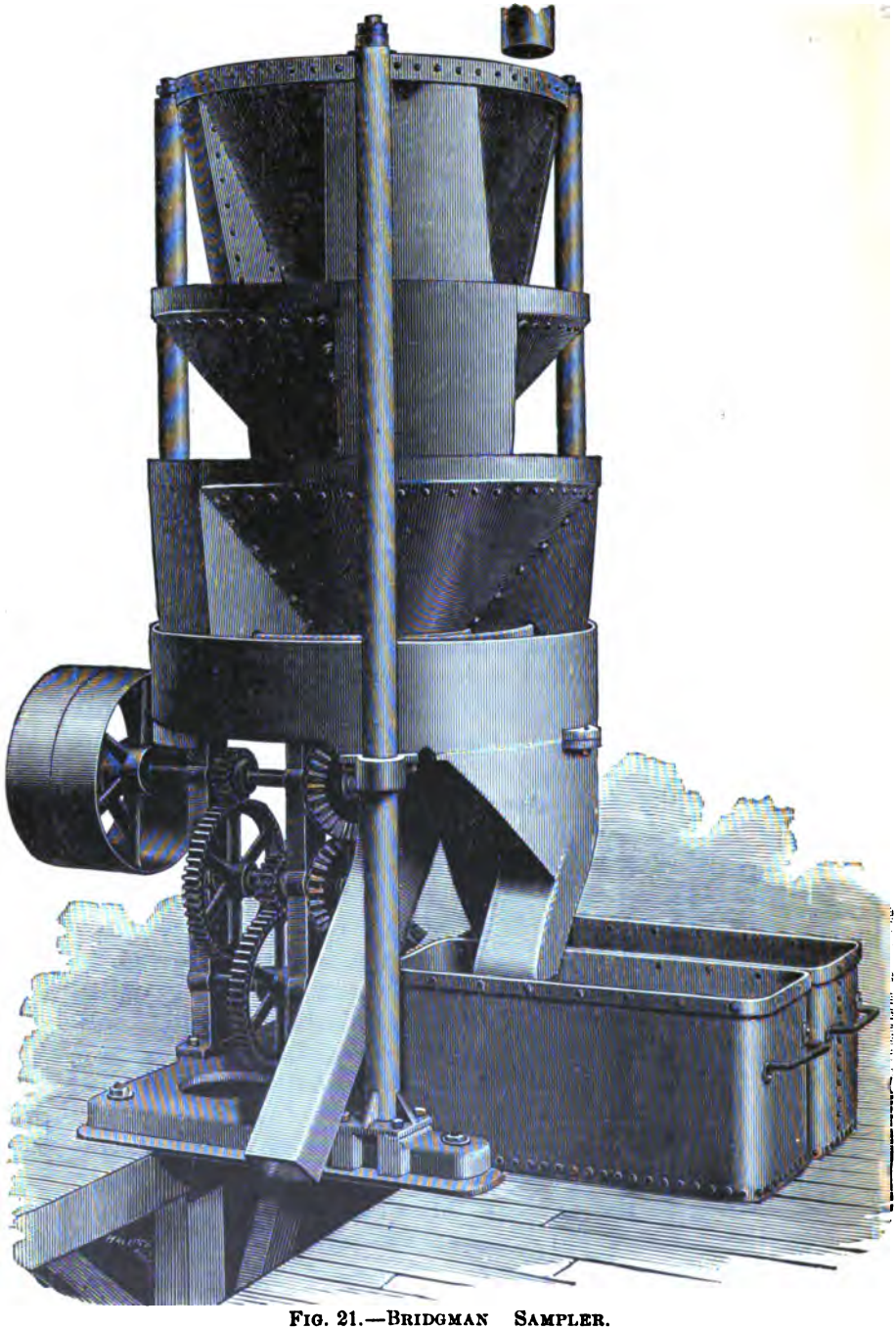


FIG. 21.—BRIDGMAN SAMPLER.

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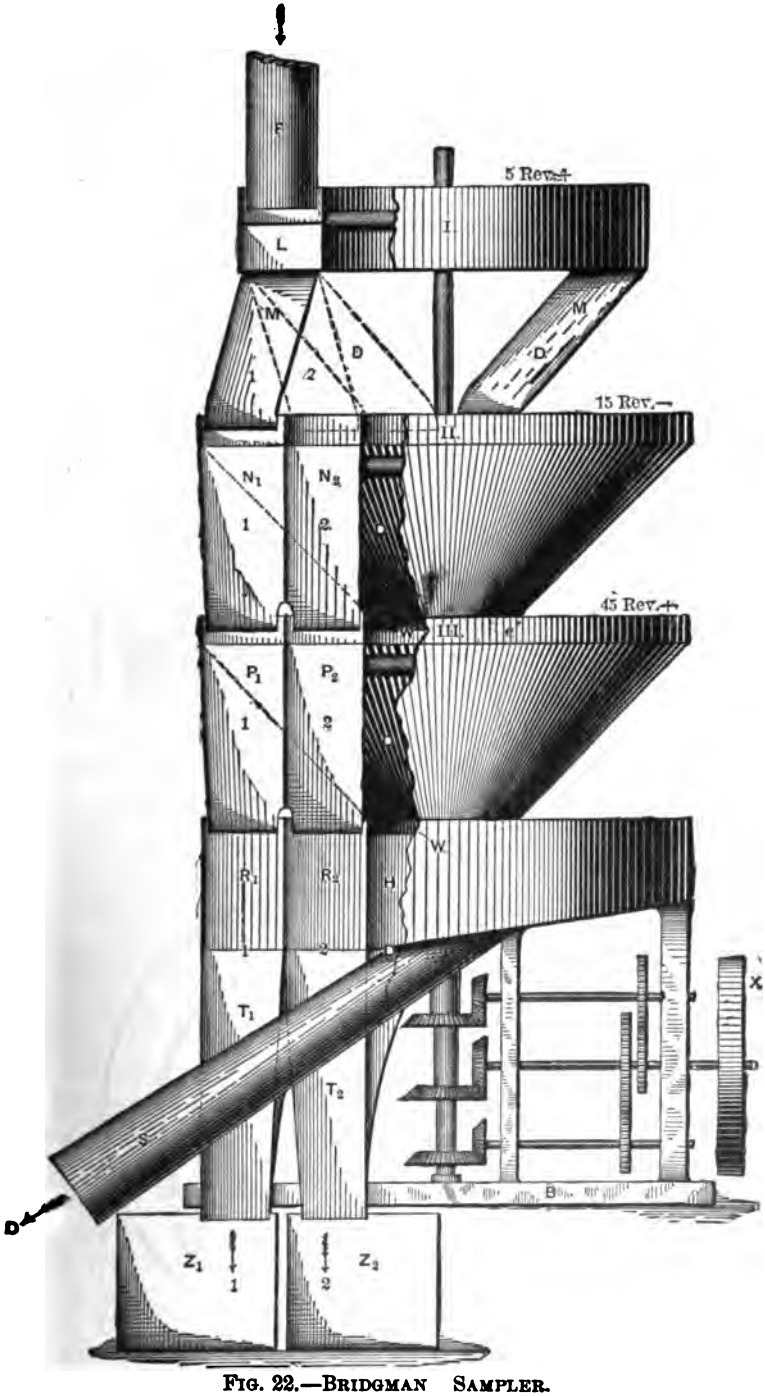


FIG. 22.—BRIDGMAN SAMPLER.

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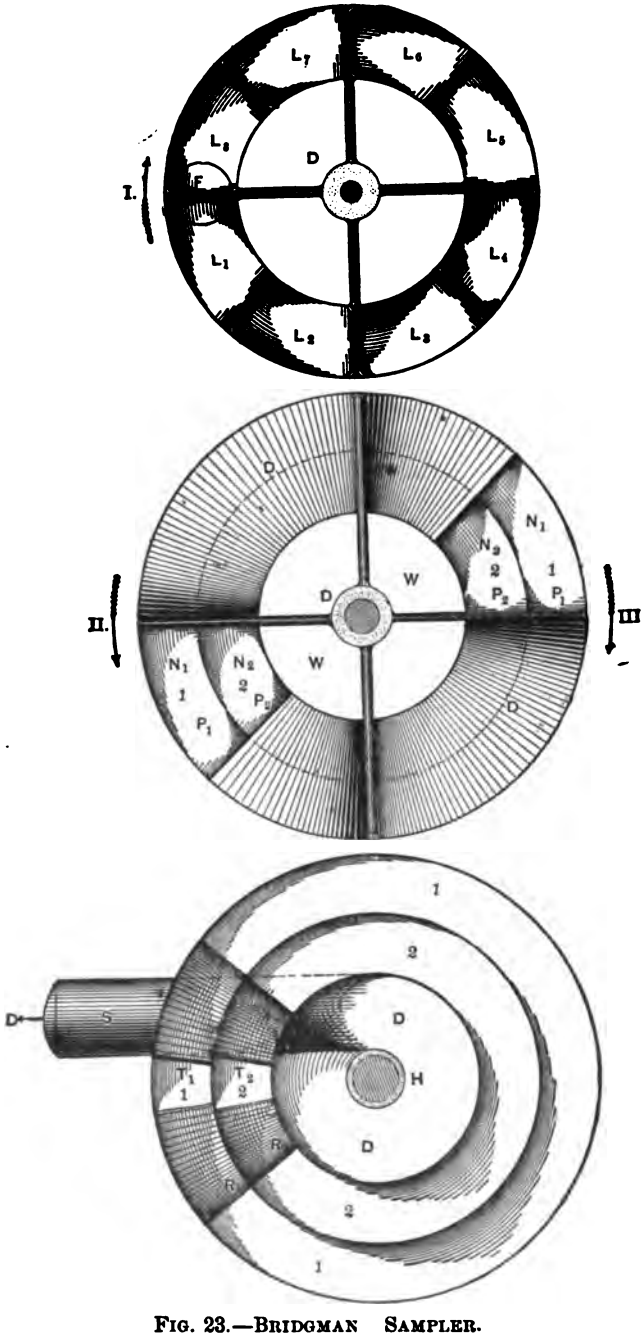


FIG. 23.—BRIDGMAN SAMPLER.

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ble spout is attached, one in the direction of *M*—1, a second in that of *M*—2, and six in that of *M*—*D*. When the apportioner is in motion, *M*—1 describes the circular path 1—1, *M*—2 the path 2—2, and the rest the path *W*—*W*. The intermediate (II.) and lowest (III.) apportioners have the same construction: a hollow truncated cone with the central discharge opening *W*, and four vertical shoots N_1—N_1 and N_2—N_2, each of which represents one-eighth of the path covered by the spouts *M*—1 and *M*—2.

If crushed ore is fed through the shoot *F* it will fall equally into all the spouts; thus, with every revolution of apportioner I., one-eighth of the ore passing through *M*—1 goes to make up the original sample, the other eighth passing through *M*—2, the duplicate sample, and the remaining six the rejected ore, which passes off through *W*, *W*, *H* (Fig. 23) and spout *S* (Fig. 22). The samples caught by *M*—1 and *M*—2 now undergo separately the same operations. To follow the one in *M*—1; it is intercepted by apportioner II., which cuts out and delivers to *D* three-fourths of it, the remaining fourth passing through shoot *N*—1 to receive the same treatment by the apportioner III. Thus the amount *a'*, cut out by *M*—1 with every revolution of apportioner I. and forming one-eighth of the ore fed during the revolution, is reduced to $\frac{a'}{4}$ by the apportioner II., and to $\frac{a'}{16}$ by the apportioner III., when it drops into the sample-box Z_1. Thus a_2 cut out by *M*—2, undergoing the same operation, is reduced in like proportion and collected in box Z_2.

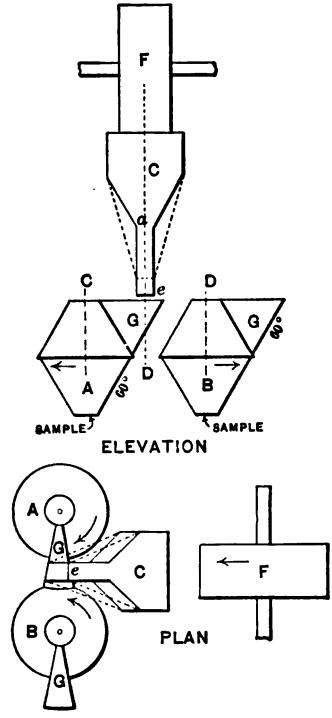
This machine was the first constructed to take out duplicate samples. It will be noticed that it cuts down the first sample twice without mixing or re-crushing. As this is only permissible with finely divided material in which the precious metals are uniformly distributed, the machine does its best work on concentrates and rolled matter. It has a great capacity, varying from 15 to 25 tons per hour, is worked cheaply, and can be easily cleaned.

Type *B*, described and illustrated in the reference given, is a smaller machine of similar construction furnishing only one sample.

The general arrangement of a sampling mill using the Bridgman sampler is shown in Figs. 159 and 160, representing a small smelting plant as built by Messrs. Fraser & Chalmers of Chicago, Ill.

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*The Vezin Sampler.**—The leading features of this sampler are shown in elevation and plan in Figs. 24 and 25. They represent two samplers, *A* and *B*, attached to spindles *C* and *D*, revolving in opposite directions. They receive at *e*, the end of the discharge spout *C*, the ore that has been raised by the elevator *F*. The sampler consists of two hollow truncated cones, joined



FIGS. 24 AND 25.—VEZIN'S SAMPLER.

at their bases. The upper cone has one or more scoops *G* of the form of a sector, which, passing through the stream of ore, cuts out the sample, and delivers it into the interior of the lower cone, whence it is conducted away through a sheet-iron pipe, while the rejected ore shoots through the space between the cones into a receiver, and thence, by another pipe, to bins or

* Private notes and "Transactions of American Institute of Mining Engineers," xxvi., p. 1098.

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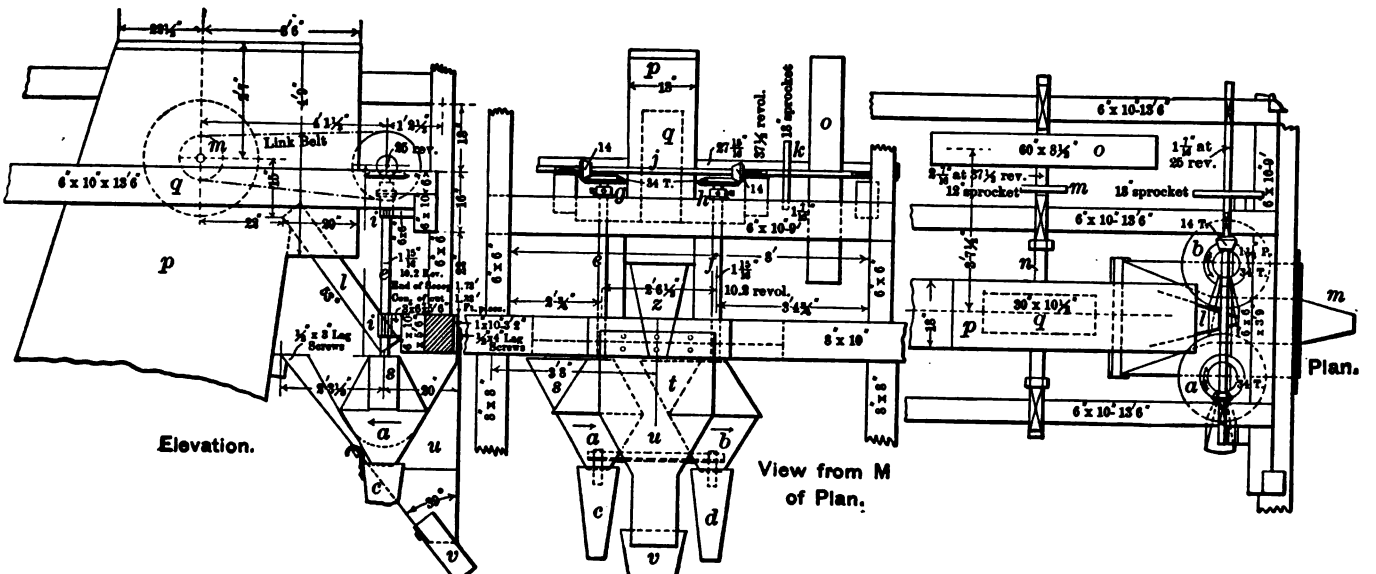
cars. The upper cone, in which are the scoops, simply serves to prevent any stray pieces from falling into the lower one and vitiating the sample. It is composed of segments, which are either blanks or have scoops in them, so that the number of these to be used can be varied to suit the requirements. The size of the scoop, and in consequence, to some degree, that of the sample itself, must be governed by the size of the largest pieces that are in the stream of ore. It should be at least four times as wide as this; the number of revolutions depends upon how frequently it is necessary to take out a scoopful. When the machine is used for taking but a single sample the number of scoops may be increased until they occupy as much as one-half of the total circumference. If arranged for taking two samples, two and even four scoops may be put upon each machine, the number depending upon the width of the scoops. Assuming that the machine makes but 10 revolutions per minute, a scoopful will be taken out once in six seconds: with ore crushed to 3 in. diameter this scoop should be at least 1 ft. wide in the center, and if it is to occupy one-eighth of the circumference of the circle it describes the diameter of this circle would have to be 30½ in. with 1-in ore; the scoop being about 5 in wide, it would take out $\frac{1}{20}$ of the ore, or, if two or four were put in, the sample would be $\frac{1}{10}$ or $\frac{1}{5}$ respectively. The sample flows into the feed hopper of a pair of rolls, and is discharged into a second elevator, which passes the ore over a second machine, by which it is cut down as much as may be considered necessary. An inclined spout *E* is preferred, as a solid stream of ore is then presented to the scoop. Assuming that the spout is inclined 58°, and the speed of the stream is 6 ft. per second, and the diameter of the circle described by the middle of the scoop 30 in., the machine making 15 revolutions, the pieces will strike the edge of the scoop, the speed of which is 2 ft. per second, at an angle of 78°, and if the speed is increased to 24½ revolutions, giving a mean speed of the scoop of 3.2 ft. per second, this angle becomes 90°; that is the relative movement is the same as if the scoop were standing still and the pieces falling vertically. The limit of speed is that at which the centrifugal force will prevent the ore from sliding down the scoop. If the first sample is so large that it can safely be cut down again, it is allowed to accumulate on the floor, or in a bin, and after it is complete it is passed into the elevator, to be again

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cut down by the same sampler. Thus, if the machine is gauged for taking $\frac{1}{4}$ or $\frac{1}{8}$, by this second cutting down, the result will be a sample of $\frac{1}{32}$ or $\frac{1}{64}$. To avoid this interruption in the crushing, one machine has been placed below the other. The scoopfuls from the upper one are passed through a revolving inclined cylinder, or upon a rapidly moving feeding shoe, so as to give a continuous stream over the lower machine. With the feeding shoe, the pieces fall but an inch as they strike the edge of the scoop. Such a feeding shoe would, on this account, be a good arrangement to deliver very coarse ore to the machine from the head of the elevator, especially if the latter is not a rapidly moving machine, and in consequence delivers the ore intermittently. The use of a single machine is preferred, on account of the simplicity of the plant, as the stoppage in running the crusher is of very short duration.

The Figs. 26, 27 and 28 represent the leading parts of the sampling works of the Metallic Extraction Co., of Florence, Colo., using the Vezin Sampler. Duplicate samples are taken by the two samplers, *a* and *b*, the samples being discharged through discharge spouts *c* and *d*. Each machine, as made, takes but $\frac{1}{16}$. The size of the sample could be increased to $\frac{1}{8}$ or $\frac{1}{4}$ each, by using two or four scoops on each of the samplers. Taking duplicates not being considered necessary, the two $\frac{1}{16}$'s are generally run together, and this gives a sample of $\frac{1}{8}$. The lower cones of these samplers are attached at their larger bases to cast-iron spiders, the hubs of which are keyed to the spindles *e* and *f*. These are suspended by collars *g* and *h*, and are kept in position by guide boxes *i*. They are rotated by beveled gears and a horizontal shaft *j*, which is set in motion by sprocket wheel *k*, connected by chain and sprocket wheel *m*, which is on the main shaft *n*, driven by main pulley *o*. The upper cone of the sampler is set upon the lower, and secured to it by small bolts. The ore is raised from the crusher discharge in the elevator *p*, the pulley of which *q* is attached to the main shaft *n*, and delivered into the inclined conveying spout *l*, made of No. 10 steel plate. The samples are taken out by the scoops *s* and *t*, while the rejected ore, passing between the two samplers, is collected by the hopper *u*, and discharged through spout *v*, and thence into a screw conveyer, which takes it to its proper bin. The two conveying spouts *c* and *d*, made of No. 16 steel (not shown), lead the sam-

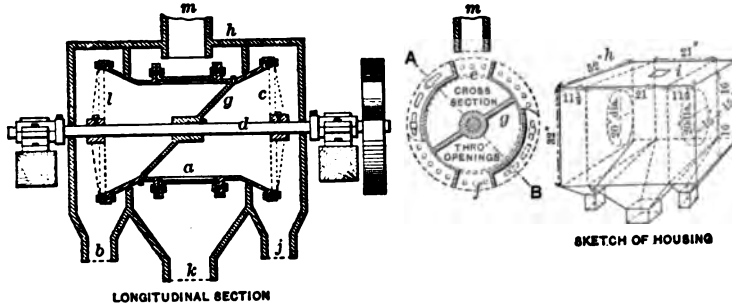
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FIGS. 26, 27 AND 28.—VEZIN SAMPLER. METALLIC EXTRACTION CO., FLORENCE, COLO.

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ples into wheelbarrows or cars, to be taken to the rolls for finer comminution. If the sample is large, it is collected on the floor, and then shoveled back into the elevator, and again passed over the machines. The drawings show that the main elevator shaft makes 37½ revolutions per minute, the horizontal shaft driving the samplers 25 revolutions, and the samplers themselves 10.2. This speed might well be increased to at least 15 revolutions. The samplers, made of No. 10 steel plate, are 24 in. and 6 in. in diameter, and 30 in. high. The angle of the upper cone is 60°, and that of the lower the same. The scoops of each sampler, also of No. 10 steel plate, form one-sixteenth of the circumference, the sides are vertical and riveted to the cones. Although it is exposed to greater wear than any other part of the machine,



FIGS. 29 TO 31.—CONSTANT DUPLEX SAMPLER.

in sampling 32,000 tons of hard Cripple Creek ore, the cutting edges were only worn down ½ in. Some of the main advantages of the machine are the simplicity of construction and the ease with which it can be cleaned should the ore be sticky.

The machine has been checked repeatedly upon itself, and with the most careful hand-sampling, and is giving complete satisfaction, as well with the Cripple Creek and other gold ore, crushed to ¼ in., as with the coarse ore which is so desirable for furnace work.

The Constant Sampler.—The form known as "Duplex Sampler" is shown in Figs. 29, 30, and 31. It consists of a cast-iron cylinder *a* attached to a horizontal shaft *d*, the ends *b* and *c* being conical. The cylinder has two slots *e* and *f*, the widths of

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which can be regulated at will. A cast-iron diaphragm *g* divides it diagonally, leaving one of the slots in each division. The sampler is inclosed by a wooden housing *h*, having one inlet *i* and three outlets *j*, *k*, and *l*. The ore fed in a continuous stream, through the shoot *m*, strikes the cylinder, which makes from 10 to 30 revolutions per minute. It is divided into three parts; the duplicate samples, which are cut out through the openings in the cylinder and discharged toward the right and left into their respective shoots *j* and *l*, and the rejected ore which falls into the central shoot *k*. The apparatus does accurate work, is small, light (weight 500 lb.), and simple in construction. It requires little power, has a large capacity, samples ore up to 2 in. in size and is easily cleaned. The machine has been adopted by many Eastern works.

A simpler form constructed on the same principle is the "Cone Sampler." It consists of a hollow cone attached to a horizontal shaft and has only one opening in the middle of the side, thus taking out a single sample discharged at the base of the cone into a shoot, while the rejected ore glides over the surface of the cone and is collected in a hopper. The machine weighs 375 lb.

Comparison of Methods.—In summarizing the advantages and disadvantages of the different methods of sampling, it may be said that with proper care an accurate sample can be obtained by any of them. Fractional selection and mechanical sampling are the best for handling large quantities of ore; small amounts are effectively sampled by quartering, fractional selection, and by the use of the split shovel. The best finishing methods are quartering, using the split-shovel, and the riffle. Finally, to compare hand sampling and machine sampling, it may be said that, while in the former you are dependent on the workman, and the method is slow, expensive, and requires a large space to work in, machine sampling has not as yet displaced it, and a smelter will be best served by not depending entirely upon either method, but using both as occasion may require.

Finishing the Sample.—This is best done in a well-lighted room, adjoining the sampling floor. The sample must be protected from the wind, from dust, and from any possibility of being tampered with. The seller may be permitted to watch the sampling through one of the windows, but not to enter the room, where the head sampler works alone. If the sample is moist it

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must be dried somewhat, so that it can be ground. Ore that is very moist or frozen when it comes to the works can generally be sampled down to one ton before it needs to be dried.

The sample obtained by one of the different methods described, weighing from 50 to 100 lb., has to be prepared for the laboratory. If larger it is ground either by hand or by machine to pass a 10-mesh sieve. If this is done by hand, a heavy muller, with head 4½ by 7 in., is used on a circular cast-iron plate of from 3 to 4 ft. in diameter and from 1 to 1½ in. thickness. The plate, the working side of which is planed, stands free, so that the sampler can walk around it while grinding. Sometimes an oblong plate is used, 18 by 24 in., with a rim ¾ in. high on two or three sides and a muller head 4 by 6 in. More commonly the comminution is effected by a machine, the so-called sample-grinder,* which resembles a coffee-mill. The sample is now reduced to about 10 lb. by quartering, using the assayer's riffle, rarely by channelling. In quartering, a funnel is very handy to form the cone, which is then stirred down with the palm of the hand, with a 2 by ¾ in. lath from 12 to 18 in. long, held horizontally, or with a spoon held vertically. The assayer's riffle resembles a small split-shovel without a handle. The other tools are a small scoop and a flat paint-brush.

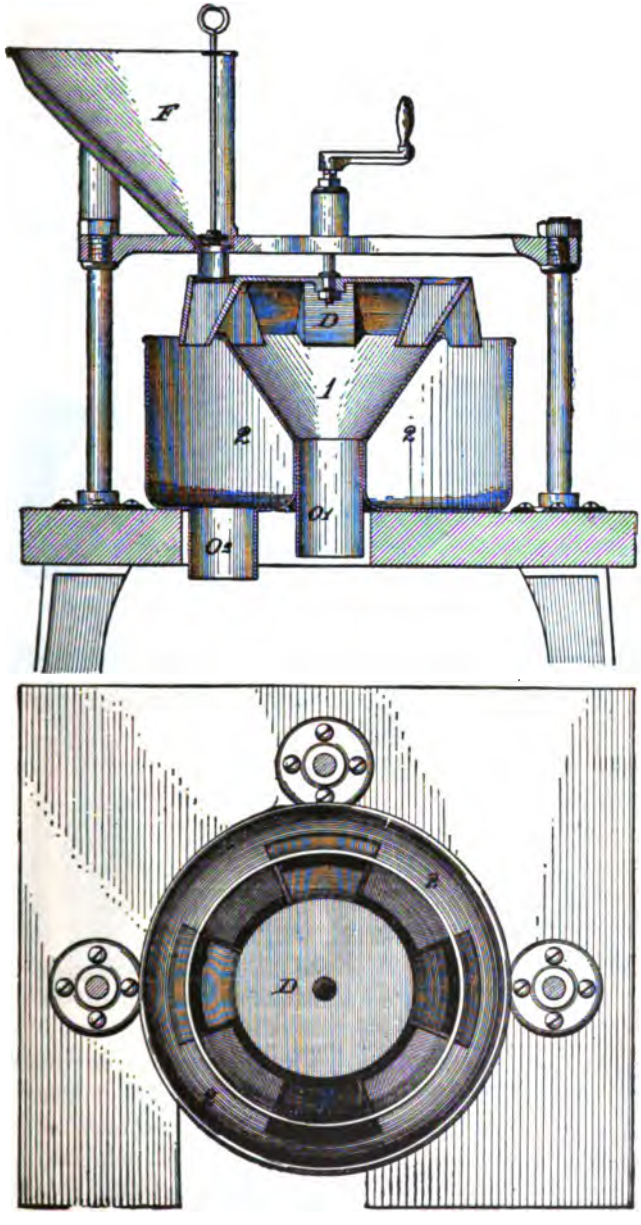
The Bridgman Laboratory Sampler† (Figs. 32 and 33) is a very handy machine to simplify the quartering, occupying a space 14 in. square and being about 14 in. high. It has as its main parts a divider *D* (which is set in motion by hand, by clockwork, or by any convenient power), a funnel 1, and a receptacle 2, with discharges O_1 and O_2 respectively. The ore filling the hopper *F* runs in a continuous stream to the divider, which gives eight cuts to the revolution: four of these, forming the sample, are delivered into the funnel; the other four, the discarded portion, are collected in the receptacle.

From the 50 or 100 lb. either a single sample may be made or two, called the original and duplicate. In the latter case the two halves of the ore obtained are treated separately; if the riffle is used, that part which has been caught by the troughs is dumped alternately to one side and the other, forming two heaps,

*The makes of Fraser & Chalmers, in Chicago, of Hendrie & Bolthoff, in Denver, and of the Engelbach Machine Co., in Leadville, are those generally used.

†"Transactions of American Institute of Mining Engineers," xx., p. 437.

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FIGS. 32 AND 33.—BRIDGMAN LABORATORY SAMPLER.

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which are then worked up separately. The duplicate sample is useful in two ways: first, to verify the work; secondly, to expose any tampering with the samples. When these are completely dried they are ground to pass a 20-mesh sieve, then well mixed by rubbing in a mortar, and cut down to about 1 lb. This is again ground on the plate to pass a 60-mesh (or more commonly an 80-mesh) sieve, and is then again well mixed by rubbing, after which it is ready to be filled into the large-necked 4 oz. sample bottles or envelopes.



FIGS. 34 AND 35.—THE BRIDGMAN MIXER AND DIVIDER.

To simplify this last operation Bridgman* has devised a mixer and divider, shown in Figs. 34 and 35. The mixer, a large funnel with movable cover, is filled with the ore, well shaken, and then passed to and fro over the divider, which discharges the sample evenly into the four bottles beneath it.

Sometimes scales of metal in the ore—"metallics"—are too large to be ground with the rest to pass the sieve. They are then

* "Transactions of American Institute of Mining Engineers," xx., p. 439.

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weighed, assayed separately, and the average result calculated by the following formula:

$$\frac{S.a + P.a'}{S + P} = a''$$

in which S=weight of scales; P=weight of pulp; a and a', the respective assays in ounces per ton; a'', the average assay. The scales are best assayed all together, as dividing them is liable to cause error.

Each sample is divided into three parts, filling three bottles or envelopes: one for the seller, one for the smelter, and one for the control. Triplicate assays are made from each sample, and thus both sample and assay are checked. If much discrepancy is found between the original and the duplicate, re-sampling becomes necessary; if only little, it is averaged.*

It is hardly necessary to say that the sampling floor and all the apparatus must be thoroughly swept and cleaned after each operation; this is especially important in the finishing-room, as the smaller the sample becomes the more carefully must it be protected. The last traces from previously comminuted ore can be best removed by using a small quantity of the new sample, taking care to return this to the general bulk. As it is almost impossible to perfectly clean the bucking boards, screens, etc., some works have several of them for the high and low grades of ore at the opposite ends of the finishing room. Rolling on oil-cloth, rubber cloth, or glazed paper, in order to mix the samples, has been given up, as it was found that fine particles of rich ore adhering to the cloth or paper could not readily be removed, and further that in rolling, the fine particles (often the richest) balled together and were not well distributed. Mixing on a glass plate or in a Wedgewood mortar is common to-day. Some plants, in order to insure accuracy, have gone so far as to have two separate sampling and assay departments.

The charges made for sampling vary from $1.50 to $2 per ton, according to the grade and character of the ore. Rich silver ores, consisting mainly of gangue and silver mineral, and gold ores with free gold, often require triplicate samples, and always closer work than uniform low-grade ores. The last are sampled, if in large quantities, at $1 a ton: and with concentrates requiring no crushing the price is often lower.

* "Leadville Smelters' Regulations," *Engineering and Mining Journal*, Oct. 27, 1883.

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§ 33. General Arrangement of Sampling Department.—The sampling department should be arranged so as to avoid all unnecessary handling of the ore.

For hand sampling the simplest way for small works is to have the ore-bins below the main sampling floor, so that the rejected ore can fall at once into the bins and remain there until required for the blast furnace. The distance must be sufficient for the ore to be discharged from the center into any of the bins. The receiving floor need only be raised enough above the sampling floor that the ore discharged by the crusher shall fall into a wheelbarrow placed below, the mouth of the crusher being on a level with the receiving floor and near the middle of the edge. A distance of 4½ ft. between receiving and sampling floors is sufficient for a No. 5 Blake crusher. The receiving floor is about a third as large as the sampling floor. The floors are made 3 in. thick and consist of two layers of planking, the upper being boards 1 in. thick, running across the heavier planks, and parallel with the mouth of the crusher to facilitate the shoveling. In front of the crusher the floor is protected by a sheet-iron plate, say ⅛ in. in thickness, fastened by countersunk screws.

Another arrangement, found in larger works, is to place the sampling-room opposite the ore-bins on the other side of the track. This is laid so low that the floor of the car is on the same plane as the sampling-room and the runways on top of the ore-bins. The sampling room is a simple, oblong building with stalls on the four sides to keep the samples. It has only one floor, on which are placed crusher and rolls. The finishing-room is partitioned off from the main room. While the ore is being unloaded from the car the sample is taken by fractional selection and wheeled to one of the stalls on the sampling floor, the bulk of the ore passing straight to the bins.

In machine sampling, which is used only in the largest works, the sampling mill is never directly connected with the smelter building. Its arrangement will depend on the character of the machinery and the configuration of the ground (see § 69).

A lot of ore, after being discharged into a bin or on a bed, is spread out evenly before another lot is received. Each subsequent lot will thus form a distinct layer over the preceding one.

At many works it is customary to collect separately the different classes of ore, such as sulphide and oxide, acid and basic,

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high grade and low grade. A bin is often made just so large that a carload of ore when discharged into it shall form a layer covering the entire floor space. In making smelting-beds of from 1,500 to 3,000 tons of ore, it is frequently not possible for a small shipment to be spread over the entire bed; in such cases small lots are "matched," *i.e.*, they are distributed in parallel strips over the ore-bed.

A special book is kept for entering the contents of the different bins and beds, the averages being calculated according to the formula in § 31.

RECORD OF BINS AND BEDS.

Year.	Bin. / Bed. No.	Date. Closed.	Date. Taken for Smelting.	Date. Smelted.	Contains Lots No.	Tot. Weight. Tons. Wet.	Tot. Weight. Tons. Dry.	Average p. c. of moisture.	Average. Oz. per ton. Au.	Average. Oz. per ton. Ag.	Average. Per cent. Pb.	Average. Per cent. Cu.	Average. Per cent. Zn.	Average. Per cent.	Average. Per cent.

Assay. Per Cent. SiO_2.	FeO.	MnO.	CaO.	MgO.	BaO.	Al_2O_3.		Remarks.

In order to check the "average" in the table, an average sample is made up of the single samples, paying due regard to the moisture, and then assayed and analyzed.

§ 34. Receiving and Sampling of Fluxes and Fuel.—Fluxes and fuel on arriving at the smelter are weighed on platform scales and unloaded as near the feed floor as possible. It is necessary to keep large quantities of fluxes always on hand. As

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they take up a great deal of room and are not injured by the weather, they are generally left outside the building. The fuel is placed in sheds for protection. The breaking of fluxes is done by hand or in a coarse-set crusher. The latter method makes more fines. The most desirable size for iron ore and limestone is that of a man's fist; for quartz, that of an egg.

In sampling fluxes and fuel, a so-called grab-sample gives sufficiently near results. This consists in taking out bits or handfuls here and there from the top, middle, and bottom of the heap until about 50 lb. of iron ore, 30 of limestone, or 10 of coke have been taken. From the grab-sample the moisture-sample is taken, and the remainder crushed fine and cut down to be analyzed in the same manner as a regular ore-sample.

§ 35. The Assaying of Lead-Silver Ores.*—This is for two purposes: first, to ascertain the amount of metal contained in the ore; second, to determine its chemical composition so far as this relates to the making up of the smelting charge.

§ 36. Purchasing of Lead-Silver Ores.†—In purchasing lead-silver ores the character of the lead-bearing mineral and the chemical composition of the gangue have to be considered. If the lead mineral is a sulphide, the ore has generally to be roasted; this is not necessary if the lead is present in the form of carbonate. Further, an ore may be either self-fluxing, acid, or basic, *i.e.*, requiring no fluxes, requiring a base (iron, manganese, lime), or requiring an acid (silica) to form a desirable slag. Silver ores are commonly acid, thus the basic ores command a higher price. In purchasing basic ores, the base paid for is iron, with its substitute, manganese. The so-called "base excess" is that amount of available iron and manganese which is obtained by adding the percentage of metallic iron to that of metallic manganese, and deducting the percentage of silica. The impurities in an ore affect its price. They may necessitate

*C. and J. J. Beringer, "A Text-Book of Assaying," London, 1897; Brown, "Manual of Assaying," Chicago, 1897; Furman, "A Manual of Practical Assaying," New York, 1896; Ricketts and Miller, "Notes on Assaying," New York, 1897; Balling, "Probirkunde," Brunswick, 1879; "Supplement," Berlin, 1887; Kerl, "Metallurgische Probirkunde," Leipsic, 1882; "Supplement," *ibid.*, 1887; Kerl, "The Assayer's Manual" (an abridged treatise of the larger work), trans. by Brannt and Wahl; new edition by Garrison, Philadelphia, 1890; Riche, "L'art de l'essayeur," Paris, 1868.

†Kirchhoff, "U. S. Geological Survey: Mineral Resources of the United States," 1883, p. 251; Furman, *School of Mines Quarterly*, xv., p. 1; *Engineering and Mining Journal*, Nov. 26, 1892.

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a preliminary roasting (sulphur, arsenic), may impair the fusibility of the slag (zinc, magnesia, baryta), or may cause loss of lead and silver by slagging or volatilization (zinc, arsenic, antimony), or finally may render the lead impure (zinc, arsenic, antimony, copper). The price paid for an ore will therefore be in inverse ratio to the percentage of impurities present.

A smelter, especially if so located as to draw its supplies from a number of mining districts, treats not only argentiferous lead ores, but extracts the precious metal also from real silver ores, called dry ores on account of their want of lead, by mixing them with ores that contain more lead than is required for the charge. Smelting can thus often compete with milling on account of the higher percentage of precious metal extracted from the ore.

In bidding for an ore deductions are made for loss and for cost of smelting, which will vary from the causes just mentioned. In the smelting charges are included the cost of shipping and refining the base bullion obtained.

The value of lead is given in units of 20 lb. to the ton of 2,000 lb. avoirdupois. Its price is regulated not simply by the fluctuations of the market, but also largely by the scarcity or abundance of available lead ore at the works. If the lead runs lower than 5% it is not paid for.

The value of silver and gold is given in troy ounces per ton of 2,000 lb. avoirdupois. The price of silver is regulated by the New York quotation of the day, that of the gold varies slightly, according to the amount present, from $19 to $20 per oz. The gold in an ore is not counted if it runs less than 0.05 oz. per ton.

Competition has forced works to pay for copper contained in sulphurets when the dry assay exceeds 2%. It is customary to deduct 1.5% from the wet determination to obtain the so-called dry assay. Sometimes such small amounts of copper are paid for as lead, the percentage of copper being added to that of lead. There are various ways of arriving at the minimum smelting charge for an ore that is offered for purchase. The following gives very satisfactory results, and can be made to suit all cases. It is based on the cost of smelting what is called a neutral ore, and then debiting and crediting the actual ore for which the smelting charge is to be fixed, as it differs in composition from the standard. The so-called neutral ore is of different composition in different districts, even differing sometimes in the same

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place. Thus Furman gives it for Denver as SiO_2 30%, Fe 30%, Pb 13%, Zn 8%, S 5%, and credits for CaO and to some extent for MgO and BaO. Another and more general statement for the same locality is that a neutral ore is one in which the insoluble residue is equal to the sum of iron (Fe) and manganese (Mn) it contains, and buying on this basis is called buying on a neutral basis, special (varying) rates being made for lime, magnesia, baryta, zinc, and sulphur. Taking Denver and Pueblo rates, f. o. b. works,* the manner of figuring may be shown by two examples:

1. A sulphide ore containing: Ag, 75 oz.; Pb, 13%; Au, 0.5 oz.; SiO_2, 25%; Fe, 35%; Zn, 12%. Debit: Smelting, $4; roasting, $2; fluxing (there is an iron excess and enough lead); zinc charge $1 (2 units at 50c.)=total, $7. Credit: Gold, $0.75 ($20.50 an oz. received by the refiner and $19 an oz. paid to the miner; silver, $1 (97% saved and 95% paid for); lead, $1.30 (worth 60c. on a $4 basis and 50c. paid for, see later); iron excess, $1 (35—25=10 units at 10c.)= total, $4.05. The minimum treatment charge that the smelter can make and come out even is $7—$4.05=$2.95. Assuming that the smelter desires about $1 a ton profit from the smelting he would make his treatment charge $4 per ton.

The value of the ore is calculated on the following basis:

Lead: Assuming the loss in lead not to exceed 10%, and deducting freight and refining charges, lead is worth 60c. per unit, when the price of lead in New York is $4 per 100 lb. Only 50c., however, is paid, which is called paying on a $4 basis. The pay will rise or fall 1c. for every corresponding variation of 5c. per 100 lb. in the New York market from the $4 basis.

Silver: 95% of the New York quotation for the day.

Gold: $19 an ounce.

The value of the ore for which the smelting charge is calculated will be per ton: Lead, 13 units at 30c.=$3.90; silver, 75 oz. at 68c.=$51, less 5% loss, or $48.45; gold, 0.5 oz. at $19.00= $9.50—total, $61.85, less a smelting charge of $4, or a net value of ore per ton of $57.85.

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*Cost of smelting, per ton, $4; cost of roasting, per ton, $2; pay for gold, per oz., $19; pay for silver per oz., 95% of New York quotation; pay for lead per unit, 10c.; charge for zinc per unit, in excess of 10%, 50c.; pay for iron or lime excess, per unit, 10c.; charge for silica excess, per unit, 10c.

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2. A dry silicious ore containing: Ag, 250 oz.; Pb, 4%; SiO_2, 60%; Fe, 10%; CaO, 12%; Zn, 6%. Debit: Smelting, $4; fluxing, $0.90 (13—4=9 units lead at 10c.); silica excess, $3.80 (60—22=38 units silica at 10c.); zinc charge, none (less than 10%)=total, $8.70. Credit: none.

The smelting charge at which this ore could be taken without making any loss would be $8.70—$5=$3.70. On such an ore, however, on account of locking up so much money, the smelter requires a larger profit than with a low-grade ore; he would make his charge about $9 per ton.

The value of the ore per ton is: Lead, 4 units, no pay; silver, 250 oz. at 68c.=$170, less 5% loss=$161.50; gold, none, less a smelting charge of $9, or a net value of ore per ton of $152.50.

With desirable lead ores carrying from 45 to 65% lead, *e.g.*, concentrates from Idaho or rich basic carbonate ores, the method of purchasing differs somewhat from the one outlined, as on account of the scarcity of lead the smelter may have to treat these ores at a loss in order to be able to make up his blast furnace mixture. He will calculate the net cost of treatment, paying 95% of the silver, and 90% of the lead, and then be guided in his offer by the less desirable ores on which he will make his profit.

§ 37. Purchasing Non-Argentiferous Ores.*—In the Mississippi Valley this is not done by bidding in the market. The "buyer" goes to the different mines and, guided by previous experience in purchasing from the same mine, offers a certain amount for the concentrated galena. The price paid for pure concentrated galena in 1896 was $32.08† per ton.

§ 38. Purchasing of Fluxes and Fuel.—In regard to the purchase of fluxes and fuel there are no general standards as there are with ores. The manner of buying and price paid vary according to local conditions.

* Clerc, *Engineering and Mining Journal*, July 4, 1885, p. 4; Blake, "Transactions of American Institute of Mining Engineers," xxii., p. 564; Spencer, "Bulletin Missouri Mining Club," i., No. 2, p. 51.

† "The Mineral Industry," v., p. 398.

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PART II.

THE METALLURGICAL TREATMENT OF LEAD ORES.

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PART II.

THE METALLURGICAL TREATMENT OF LEAD ORES.*

§ 39. Classification of Methods.—Lead ores are treated exclusively in the dry way. If free from silver the resulting lead goes to the market after it has been purified by liquating and poling. If the ore is argentiferous the silver passes for the most part into the lead (base bullion), which has then to be desilverized (Part III.). Wet methods† and electrolytic‡ extraction have been tried, but so far without success. The smelting of lead ores is carried on in furnaces of various forms. They may be classified under three heads:

The Reverberatory Furnace.

The Hearth Furnace.

The Shaft Furnace.

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*Eissler, "Metallurgy of Argentiferous Lead," London, 189–; Percy, "Metallurgy of Lead," London, 1870; Phillips-Bauerman, "Elements of Metallurgy," London, Philadelphia, 1891, pp. 640–712, Collins, "Metallurgy of Lead," London, 1899; Balling, "Metallhüttenkunde," Berlin, 1885, pp. 49–165; Kerl, "Metallhüttenkunde," Leipsic, 1881, pp. 1–128; Schnabel (Louis), "Handbook of Metallurgy," 1898, I; pp. 275–455; Stölzel, "Die Metallurgie," Brunswick, 1863–66, pp. 851–1050; Rivot, "Traité de métallurgie," vol. ii., Paris, 1872; Grüner, "Sur l'état actuel de la métallurgie du plomb," *Annales des Mines*, 1868, xiii., p. 325; also as pamphlet: Dunod, Paris; Grüner, "Notes additionelles sur l'état actuel de la métallurgie du plomb," *Annales des Mines*, 1869, xv., p. 519: also as pamphlet: Dunod, Paris; Cahen, "La métallurgie du plomb," Paris and Liége, 1863; In press, Lodin, "Métallurgie du plomb et de l'étain," Dunod, Paris.

†*Berg- und Hüttenmännische Zeitung*, 1880, p. 1 (Schaffner), p. 2 (Maxwell); *Wagner's Jahresberichte*, 1877, p. 151 (Meunier).

‡*Berg- und Hüttenmännische Zeitung*, 1883, p. 252 (Kiliani), p. 378 (Blast-Miest).

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CHAPTER VI.

SMELTING IN THE REVERBERATORY FURNACE.

§ 40. Introductory Remarks.—The process carried on in the reverberatory furnace is the Roasting and Reaction Process (§ 9). The Precipitation Process (§ 7) was formerly used with raw sulphide ores at Vienne* (France) and at Chicago,† and with roasted ores at Par and Point‡ (Cornwall), but has now become obsolete on account of the high cost and loss of metal.

The roasting and reaction process in the reverberatory furnace consists of two operations, one following closely upon the other, and both being repeated several times.

1. *Oxidation.*—The ore, crushed fine enough to pass a 4 or 5-mesh sieve, is spread in a thin layer over the hearth of the furnace and is heated gradually to an incipient red heat (500 to 600° C.). The roasting is carried on in such a way that only a part of the lead sulphide is converted into oxide and sulphate, the rest remaining undecomposed. The temperature at which this roasting is carried on and the time given to it depend on the character of the ore. Pure galena requires a low temperature to avoid agglommeration; if pyrite, blende or calcite are present the roasting can be accelerated; the lower the temperature, the more sulphate will be formed. During the operation the fire on the grate is kept low and open and the charge is raked frequently, to expose as much of the ore as possible to the action of air and heat and to prevent agglomerating, which obstructs oxidation. (For details of roasting, see § 55.)

2. *Reduction.*—The second operation is that of raising the temperature, so that the oxygen compounds may react on unchanged sulphide. The resulting lead runs down the inclined

* Kerl, "Grundriss der Metallhüttenkunde," 1881, p. 24.
† "Transactions of American Institute of Mining Engineers," ii., p. 279 (Jernegan).
‡ Percy, "Lead," p. 257.

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hearth and collects in a basin, the sulphur dioxide passes off through the flue, and the residue remains on the hearth. The temperature during the reduction period must be low, so that the charge may be only in a pasty condition, as the reactions are very imperfect, if the ore is melted. By filling the grate well up with fuel the required temperature is obtained and unconsumed air excluded. The charge is stirred at intervals to bring sulphide and oxide constituents into intimate contact.

As it is not possible to roast a large amount of lead ore uniformly in one operation, the first reaction that takes place on raising the temperature will not extract all the lead. The resulting pasty residue will be rich, consisting mainly of lead sulphide with some oxide, sulphate, silicate, and gangue. The temperature is lowered and air is admitted. A second roasting takes place and is followed by a second reacting. It takes several repetitions of the process to extract the bulk of the lead. With each one the temperature must be slightly raised as the amount of lead diminishes. To counteract the melting of the charge, slacked lime is added, which acts mechanically by rendering the charge less fusible and more spongy. It also assists the process chemically by liberating the lead (§ 7) and by decomposing the sulphide, thus helping the silver in the residue to pass into the lead. Toward the end of the process there will not be enough lead sulphide left to react on the excess of lead sulphate (§ 9) and lead oxide. To reduce these to sulphide and metal, and to make the charge more porous, charcoal or coal is mixed with it; then the roasting and reacting can proceed again.

Each successive operation will be of shorter duration than the preceding one and the lead each time a little less rich in silver. The first lead may contain four times as much silver as the last.*

The products of smelting in the reverberatory furnace are:

1. *Lead*, holding in suspension particles of ore and other solid matter, which are removed by liquating and poling (§ 115). If the ore contains arsenic, antimony, or copper, some of these elements combine with the lead and have to be removed by refining (§ 112).

2. *Gray Slag*, a more or less matted mixture of lead, lead sulphide, oxide, sulphate, silicate, gangue, cinders, and lime in

* *Berg- und Hüttenmännische Zeitung*. 1860, p. 59; 1863, p. 285 (Fallize); 1871, p. 153 (Bouhy). *Zeitschrift für Berg-, Hütten- und Salinen-Wesen*, xiv., p. 239 (Teichmann).

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varying proportions. Its tenor in lead and silver depends on the character of the ore and on the extent to which the residue has been treated in the furnace. In some cases it is crushed and washed to save only the metallic lead; in others, especially with silver-bearing ores, it is smelted in the blast furnace.

3. *Fluedust*, composed of particles of unchanged or oxidized ore, volatized lead that has been converted into oxide, carbonate and sulphate, and of fuel. If the ore contains blende, oxidized zinc compounds will also be found. The amount of fluedust formed will vary with the temperature at which the roasting and reacting operations have been carried on, and also with the skill of the furnace-man in manipulating the furnace and the charge. As it consists principally of oxidized compounds, it is worked in with subsequent charges and shortens the time required for roasting. If very impure, *e.g.*, from arsenic and antimony, it is smelted in the blast furnace (§ 54) with the gray slag. The resulting lead is hard and has to be refined (§ 112).

4. *Hearth Bottom*, consisting of hearth material soaked to some depth with metal. It is worked up in the same manner as the residues.

The roasting and reaction process in the reverberatory furnace has the following advantages: the ore is treated in the raw state, the apparatus is inexpensive, inferior raw fuel is used, hardly any fluxes are required, the bulk of the metal in a pure state is quickly extracted at a low temperature with little loss by volatilization, and if the ore is argentiferous the larger part of the silver follows the lead, and only a small quantity is left in the residue, which is either thrown away or treated at a higher temperature in the blast furnace. The great disadvantage of the method is that it is very much limited by the character of the ore. To be suited for the reverberatory furnace an ore must be a rich galena or a mixture of galena with carbonate (the former prevailing), that does not contain less than 58% of lead; 70% being a common figure. It may not contain more than 4 or 5% silica; and the non-silicious associated minerals, such as blende, pyrite, chalcopyrite, calcspar, and barite, may be present only in small quantities. The process requires much fuel and many hours of skilled labor per ton of ore treated.

§ 41. Influence of Foreign Matter.—The quantity and quality of lead that can be obtained from a given lead ore will depend

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largely on the nature and proportion of the other constituents. These may be silica and argillaceous matter, oxides of iron, limestone (dolomite), barite, fluorspar, pyrite, chalcopyrite, blende, antimony, arsenic, silver (gold), and oxide lead ores.

Silica and Argillaceous Matter have an injurious influence in both stages of the process on account of their readiness to combine with lead oxide (§ 6). It has been found by experiment that with more than 5% of silica an ore cannot be treated by the roasting and reaction process. But even such a small amount as 0.5% makes itself felt by coating particles of ore with the silicate that has been formed, this preventing the action of the air during the first period and obstructing the reactions when the temperature is raised.

Oxides of Iron.—Siderite is sometimes found with galena ores, but most of it can be removed by dressing the ore before smelting. The small amounts which remain with the galena quickly lose their carbon dioxide during roasting, and the resulting magnetic or ferric oxide acts as a stiffening ingredient while the lead is being extracted.

Limestone (dolomite) acts on the whole advantageously during the entire process, as it hinders the fusing of the charge. Any chemical action it may have is so slight that it can be regarded as practically inert matter. It loses some of its carbon dioxide and is to a small extent converted into sulphate. Like all indifferent substances it will retard somewhat the roasting by preventing the air from having free access to the particles of galena and obstructing the reactions by interfering with the necessary intimate contact of sulphide and oxide. The highest allowable amount is 12%.

Barite remains practically unchanged in the reverberatory.

Fluorspar also remains unchanged. If fluorspar and barite are present together they may increase the fusibility of the charge by combining with lead sulphate.

Pyrite is beneficial in the first stage. It favors the formation of lead sulphate (§ 9) and assists the oxidation of galena through sulphur trioxide liberated by the decomposition of ferrous or ferric sulphate. Small quantities of pyrite have also a favorable effect during the reaction period, the ferric oxide making the charge less fusible. If present to a considerable amount, say

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from 10 to 12%,* too much iron sulphide remains in the charge and is liable to form double compounds with the lead sulphide. With from 35 to 40% of pyrite† the reverberatory process has to stop.

Chalcopyrite has no special effect during the roasting, and behaves in a manner similar to that of iron pyrite in the reaction period, with this addition, that some of the copper enters the lead and impairs its quality.

Blende.—During the roasting period blende is partly converted into oxide and sulphate, and the latter perhaps somewhat decomposed, but much blende remains unaltered. From 4 to 5%‡ assists the roasting; 10 or 12% prolongs it greatly and reduces the output of lead; with from 20 to 24%§ very little lead is obtained; and from 35 to 40% stops the process. The loss in silver with blende-bearing ores is principally a mechanical loss that takes place during the roasting period.

Antimony.—This occurs with some galena ores as a simple or multiple sulphide. It has a deleterious effect even if present in small quantities of 2 or 3%. It is readily fusible and causes caking of the ore. The sulphide and the oxide are both volatile, thus causing loss. The oxide also combines readily with lead oxide, which is retained to a great extent by the gray slag; further sulphide and oxide of antimony react upon each other, giving metallic antimony, which is in part volatilized, causing again loss, and the remaining metal finally combines with the lead, making this hard. Thus antimony is probably the worst metal that can be associated with the lead.

Arsenic.—The pyrite found with lead ore sometimes contains arsenic. Next to antimony, arsenic is the worst enemy of the process. It causes losses: by volatilization, through the arsenious oxide formed in roasting; by slagging, through the combination of arsenic oxide with lead oxide; and finally by the reduction of both to metal. This combines with the lead and impairs its quality.

Silver and Gold.—Most of the silver of galena ores passes into the lead; gold behaves in a similar way.

Oxide Lead Ores, such as anglesite and cerussite, assist the operation, as they reduce the time of roasting.

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* Bouhy, *Annales des Mines*, 1870, xvii., p. 179.
† Ibid., p. 178.
‡ Ibid.
§ Rivot, "Métallurgie du plomb," p. 46.

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§ 42. Classification of Reverberatory Methods.—The practice of the roasting and reaction process varies in different smelting works. At some the principal stress is laid on extracting as much lead as possible in the reverberatory, while others aim to obtain only the major part of the lead in the reverberatory and to extract the rest from the rich residue by smelting it in the blast furnace; they thus recover a larger percentage of lead. Then, some establishments roast the ore slowly at a low temperature, which is advisable for the recovery of a large percentage of lead, while others hasten roasting by raising the temperature quickly to the permissible limit, the aim being to save labor and time, which is done at the expense of the lead. As regards the form and size of furnaces and the position of the lead-well there are also characteristic differences.

The reverberatory furnace practice may be classed under three main heads: 1. The Carinthian Method; 2. The English Method; 3. The Silesian Method.

§ 43. The Carinthian Method.—The characteristics of this method are the smallness of the charge, the slow roasting, so that for every part of lead sulphide one part of sulphate and at least two of oxide are formed, the low temperature at which all the operations are carried on, and the aim to extract all the lead in the reverberatory. The hearth is inclined toward the flue and the lead is collected outside of the furnace.

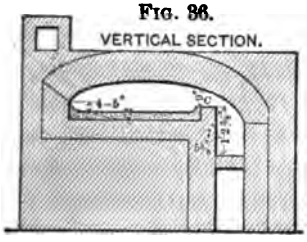
(a). *Lead-Smelting at Raibl,* Carinthia.*—The ore worked is galena (partly coarse, with from 72 to 75% of lead, and partly fine, with from 67 to 73% of lead); in exceptional cases the lead contents go as low as 58%. The other constituents of the ore are anglesite, cerussite, wulfenite, blende, willemite, calcite, fluorite, and asbestos, but these form only a very small quantity. The following analyses by Philips† in 1845 show the composition of low-grade coarse and fine ore:

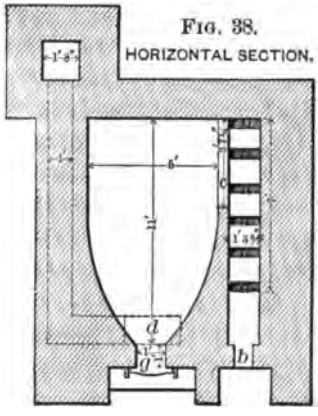
	PbS.	$PbCO_3$.	ZnS.	Sb_2S_3.	$CaCO_3$.	Insol.	Total.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Coarse.....	76.6	4.0	13.2	0.2	4.6	0.4	99.0
Fine.....	73.0	4.0	13.0	0.2	5.0	0.4	95.6

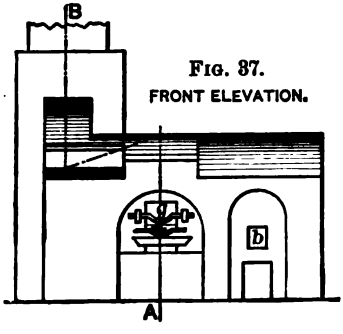
* Thum, *Berg- und Hüttenmännische Zeitung*, 1863, p. 196; "Official Report," *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1889, p. 297.

† *Annales des Mines*, 1845, viii., p. 293.

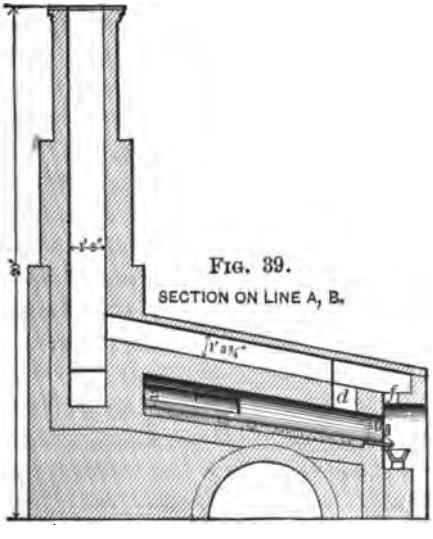
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Figs. 36 to 39.—Reverberatory Furnace at Raibl, Carinthia.

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The drawings (Figs. 36 to 39) of the furnace show an inclined hearth (9½°) with only one working-door, *g*, below the flue *d*. On the same side is the door *b* leading to the fireplace. The grate, slightly more inclined than the hearth, is parallel to the long axis of the furnace. It is built of stone and has six transverse openings. The firebridge is at *c*; the opening *f* carries off any lead fumes into the main flue. The hearth terminates at its lower end in a cast-iron gutter, over which the lead runs into the mold. Fig. 37 shows the support for the working tools. The furnace is built of sandstone and ordinary red brick; the working bottom, which is renewed every four or five weeks, is made by tamping down firmly fireclay, probably a mixture of raw and burnt clay. It is made impervious to lead by glazing with gray slag. The heating up of the furnace, leading to the fritting of the slag, is done slowly to prevent the cracking of the tamping. Furnaces are usually built in pairs, being placed side by side. They last from five to six years. The fuel used is cordwood. Dimensions are given on the drawings and in § 46.

The mode of operation is as follows: the furnace, barely red-hot from a previous charge, is repaired, if necessary, and the charge of about 400 lb. introduced through the working-door *g*, and spread out over the upper part of the hearth near the bridge to a thickness of from 1⅛ to 1½ in. No fresh fire is made, the heat of the furnace and the glowing fuel from the previous charge furnishing sufficient heat for the first slow roasting. The ore, containing a small amount of blende, is raked every quarter of an hour. With pure galena the raking is not repeated so frequently, as the quick oxidation would liberate sufficient heat to make the ore cake. The beginning of this is recognized by its adhering to the rake. The roasting period has come to an end when the blue sulphur-flames disappear, drops of lead are seen near the front end of the hearth, and the ore feels sandy. The roasting period lasts three hours, during which time the charge is rabbled from eight to nine times and from five to six sticks of cordwood are consumed. The temperature is then raised and kept pretty uniform throughout the reaction which sets in. The attendant works his charge once every quarter or half-hour, and raises the temperature slightly when the flow of lead ceases. This second period lasts from five to six hours, consumes from 16 to 18 sticks of wood, and furnishes the first half of the lead,

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which, on account of its freedom from impurities, is called virgin lead. The attendant now stops firing until he has collected the residue from all parts of the hearth into one heap. He then takes a few shovelfuls of ashes and breeze from the ash-pit, throws it on the heap of residue, and works it in in order to remove lead and to reduce lead oxide and sulphate. He then urges the fire as quickly as he can, and the so-called third period of the process, that of slag reduction, has begun. The further manipulations are the working of the residue and the stirring-in of breeze of charcoal, until after three hours the rest of the lead, the "slag-lead," has been extracted. This has to be liquated before it can be marketed.

The practice is varied in some works by raking out the residue after the ashes and breeze have been stirred in and introducing a new charge. This is worked in the usual way. The residue from this second charge is not withdrawn, but that from the first added and both worked together for slag-lead. In this case the reduction of the slag occupies from seven to eight hours.

The final residue is withdrawn from the furnace and sorted into gray slag with 4% of lead, which is thrown away, and a product to be crushed and concentrated; the heads, which assay from 50 to 60% of lead, going back to the furnace in one of the subsequent charges of the residue.

There is one furnace-man, working 24 hours, who has a helper during the day (12 hours).

Tabulated results are given in § 46. Of the products no satisfactory analyses exist except of the lead.

	Virgin Lead.		Slag Lead.	
	a^1	b^2	a^1	b^2
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Cu	0.00069	0.00010	0.00075	0.00100
Ag	0.00025	0.00008	0.00025	0.00008
Fe	0.00055	0.00700	0.00088	0.00770
Ni	.....	.....	Trace.	.....
Zn	0.00076	.....	0.00082	.....
S	0.01476	0.00400	0.01785	0.00130
Sb	Trace.	0.05700	0.00708	0.14340
As	Trace.	.....	0.00721	0.01920

[1] *Oesterreichisches Jahrbuch*, vol. xxii., p. 389; [2] *Ibid.*, vol. xxvii., p. 188.

In 1893 the furnace and method were at last given up.* The

* *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1893, p. 283.

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Belgian reverberatory furnace is now in use and is worked by the Silesian method (see § 45).

(b) *Lead Smelting at Engis, Belgium.**—The method of working at Engis differs in some respects from that at Raibl. The ores are pure. They contain, according to an average of several analyses: PbS, 93.56%; ZnS, 3.74%; FeS_2, 2.31%; $CaCO_3$, 0.35%; and traces of silver. They are free from arsenic and antimony.

The furnace has the ordinary form of a reverberatory. The hearth, widest in the middle, is slightly contracted at the bridge and narrows down considerably at the flue. The furnace has two doors, one at the side and one beneath the flue, below which there is a small kettle with a separate fireplace to receive the lead. The fuel used is bituminous coal. The furnace bottom, oval in cross-section, begins at the top of the bridge, where it is 2 ft. $8\frac{3}{4}$ in. thick, and is inclined toward the flue, so that no lead is collected in the furnace. At the flue the thickness is 1 ft. $5\frac{3}{4}$ in. The furnace has a brasque working-bottom. The brasque consists of two parts by volume of ordinary brick clay and one of coke ground fine enough to pass a 4-mesh screen. Old bottoms, containing usually about 2 or 3% of lead, are ground up and mixed with new brasque. All the brasque needed for one furnace bottom (about 4,000 lb. of clay and 1,600 lb. of coke) is prepared by two men in 24 hours. In addition to the ordinary test for the correct amount of moisture (that the brasque, when squeezed in the hand, shall cohere into a lump, but not have sufficient moisture to adhere to the hand) another one is used, that of throwing with force a ball of brasque against the wall, to which it should adhere.

In tamping, a layer of brasque 8 in. thick is first spread out evenly on the brick hearth and tamped down to 1 in.; the second 8-in. layer is not rammed down so firmly, being reduced only to $2\frac{3}{8}$ in.; the subsequent layers are made by using smaller amounts of brasque, as when spread out they are only about 4 or 5 in. thick. The bottom is not impervious to lead, which filters through to a slight extent, collects on top of the underlying red brick, and also passes into the joints. A bottom lasts about six weeks. It wears off quickly during the first two weeks, but then resists pretty well abrasion by the tools and chemical action.

* Bouhy, *Annales des Mines*, 1870, xvii., p. 159; reprint, *La fabrication du plomb*, Dunod, Paris, 1870; also in *Berg- und Hüttenmännische Zeitung*, 1870, p. 381; 1871, p. 53 ff.

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In starting a furnace with a new bottom the warming lasts six hours, slight cracks that may form being closed with brasque. Then a small charge of 220 lb. of low-grade ore with from 45 to 48% of lead is spread over the hearth and the temperature raised gradually for five and a quarter hours, the ore being raked from eight to ten times. The brasque becomes superficially soaked with lead and coated with a mixture of various more or less melted lead compounds. There result from this charge 4 lb. of lead, 5 lb. of rich slag-like material, and 130 lb. of matte-like material containing 48% of lead. The charges are increased in weight and richer ore is worked until after the third day the furnace can do economical work.

The method of working is the same as at Raibl, with the exception that all the lead is not extracted in the reverberatory. The residue, forming 12% of the original charge and assaying from 17 to 20% lead, is smelted with puddle-cinder in a small shaft-furnace. One furnace-man with a helper works a charge in 12 hours. Tabulated dimensions and results are given in § 46.

The subjoined analyses show the composition of the residue:

SiO_2.	PbO.	ZnO.	$(FeAl)_2O_3$.	CaO.	S.	Total.
Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
29.86	25.50	25.33	15.08	3.70	0.38	99.80
28.10	32.80	20.80	14.70	3.20	0.10	99.70

(c) *Lead Smelting in the Air Furnace.**—The roasting and reaction process, as carried out in the so-called air-furnace, is given here as the American improvement of the Raibl furnace.

The galena ore from the Mississippi Valley is concentrated to a high-grade pure product ranging from 70 to 84% in lead.

The furnace has a peculiar construction. The drawings (Figs. 40 and 41) show an inclined hearth *c*, with *e* as charging and working door, and *f* as discharging and cleaning door; beneath

* Williams, "Geological Survey of Missouri: Industrial Report," 1877, pp. 8-101; "Transactions of American Institute of Mining Engineers," v., p. 314; Broadhead, "Geological Survey of Missouri," 1874, p. 492. The furnace has been used in the Mississippi Valley, where it originated, to a considerable extent; but even 10 years ago it had given way largely to the ore hearth, as this has about three times the capacity, although it saves a smaller amount of lead. The percentage of lead ore worked in 1880 by different furnaces in the Mississippi Valley, as reported by the "Tenth Census of the United States," vol. xv., p. 818, was as follows: Air furnace, 10.61%; Flintshire furnace, 7.64%; Scotch hearth, 62.47%; blast furnace, 19.28%.

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is a small kettle *g*, into which the lead flows as it is set free in the furnace. The chimney is a sheet-iron pipe. Thirteen inches from the front—*i.e.*, the lower or pot end of the furnace—is the

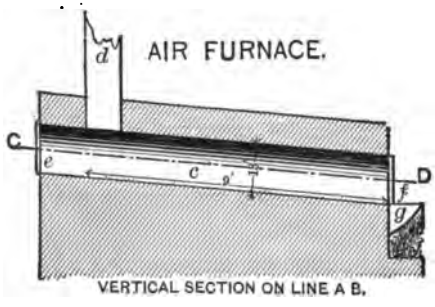


FIG. 40.

firebridge *b*, with fireplace *a* at right angles to the axis of the furnace. The bottom of the furnace is a cast-iron plate with 6 in. of gray slag or residue melted upon it. Cordwood is used as fuel.

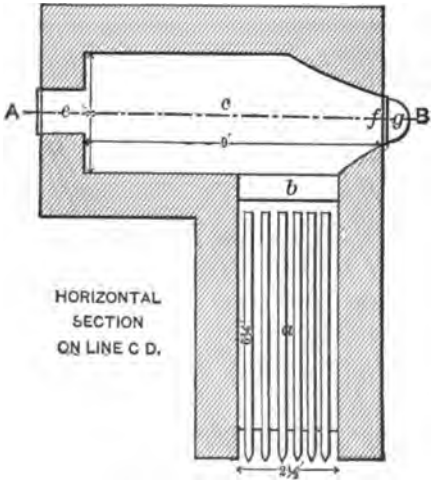


FIG. 41.

The cost of a furnace, including shed, is estimated by Broadhead at $550.

The charge, from 1,400 to 1,600 lb. of galena of from pea size to hazelnut size, is introduced through door *e* and spread evenly

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over the hearth. It is roasted for from one to one and a half hours at a low temperature, the shorter time being sufficient if the galena contains some oxidized ore. During the roasting the charge is constantly raked and moved from firebridge to flue, and from the cooler part of the furnace to the hotter, in order to heat and roast it as uniformly as possible. When this is accomplished the heat is raised and lead begins to flow. Ashes and breeze are used as stiffening ingredients. The charge is rabbled at intervals. When no more lead appears (after from seven to eight hours) the residue is drawn without any attempt to extract slag-lead in the furnace as at Raibl.

One furnace-man with a helper works in 12 hours a charge which consumes one and one-half cords of wood. About the yield nothing is known, as the contents of ore charged is not determined. Tabulated dimensions and results are given in § 46.

The following two analyses by Williams* show the composition of residue and poled lead from the Granby works:

Residue: SiO_2, 21.396%; CaO, 4.650%; MgO, 3.948%; Fe_2O_3, 3.680%; Al_2O_3, 0.152%; ZnO, 7.146%; $PbSO_4$, 2.349%; PbS, 20.929%; PbO, 34.914%—total, 99.063%. The sample yielded 3.52% of metallic lead, which makes the total metallic lead 54.82%.

Lead: As, 0.01122%; Sb, 0.00077%; Ag, 0.00080%; Cu, 0.05091%; Fe, 0.01582%; Zn, 0.00090%; Ni, 0.00281%; Pb (by difference), 99.91777%—total, 100%.

§ 44. The English Method.—The characteristics of this method are a large charge, a quick roast (with the result that for every part of lead sulphate formed there shall remain two parts of lead sulphide unchanged), a high temperature throughout, and the aim to extract all the lead in the reverberatory. The hearth inclines toward the middle of one of the sides; the lead collects in the furnace, and is tapped at intervals into an outside kettle.

Lead Smelting at Stiperstones† (*Shropshire*).—The ore is a concentrated galena with some carbonate and blende, assaying 77.5% of lead, the little gangue being principally carbonate of lime.

* "Transactions of American Institute of Mining Engineers," v., pp. 320-324.

† Moissenet, *Annales des Mines*, 1862, vol. i., p. 45; reprint, "Traitement de la galène au four gallois," Dunod, Paris, 1862; also in *Berg- und Hüttenmännische Zeitung*, 1863, pp. 243, 251, 261, 265.

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The construction of the furnace is given in detail in Figs. 42 to 47. The horizontal section (Fig. 42) shows the usual trapezoidal form of the English reverberatory with its three working doors *k* on either side, the well *b* at the front, and the fire-opening *t* at the back. It is to be noted that centers of hearth *a* and fireplace *g* are not opposite each other, the latter, with fire-bridge *h*, being 10 in. further back. The object of this is to draw away the flame from the well *b*; for the same reason the flue *d*, near the back, is made 1 in. wider than flue *d*, while both have

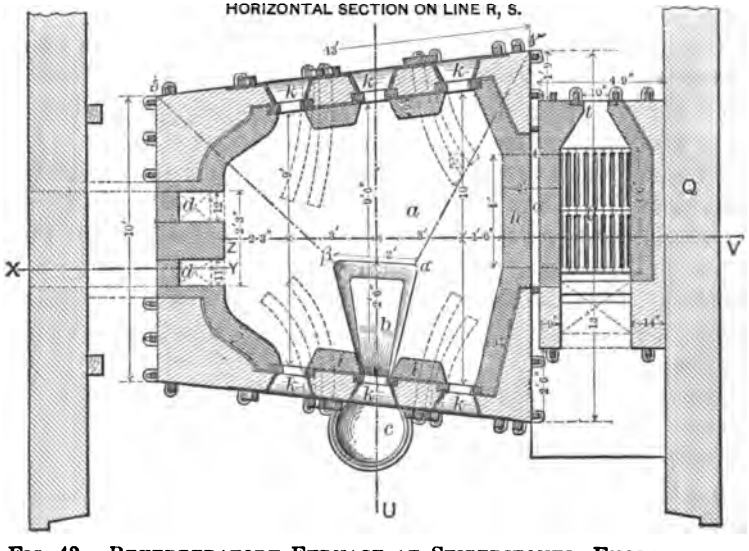


FIG. 42.—REVERBERATORY FURNACE AT STIPERSTONES, ENGLAND.

the same height. Each has its own damper *w* and *w'*. The cross-section, Fig. 44, shows the inclination of the hearth from the back to the front (24°), where well *b* is placed. This has nearly vertical walls and (Fig. 42) extends 3½ ft. into the furnace, where it is 2 ft. wide, narrowing down to a few inches in the front. The lead collected in the well is tapped through tapping-hole *m* into the cast-iron kettle or pot *c*. The longitudinal section (Fig. 43) shows the inclination of the hearth, which is 30° from the bridge and 23° from the flue. The roof *P* slopes in a straight line from top of grate *g* to flues *d d'*. The gases from the grate pass over firebridge *h* to the hearth and

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thence through flues *d d'* into flue *e*. In the front elevation (Fig. 45) can be seen the chimney *f*, which is 2 ft. 6 in. by 2 ft. in the clear and 20 ft. high from the ground. It does not communicate with the interior of the fireplace. It carries off gases that would enter the smelter building when the fire is fed or stoked or when the doors are open, and also vapor that comes from the ash-pit *u* (Fig. 43), which is filled with water.

The following details will elucidate the drawings. In Figs. 42, 43, and 45 one end of the furnace is seen to be built against the wall *Q* of the building. At the others there is a passage *D*; 15 ft. of free space at front and back give sufficient room for working the furnace.

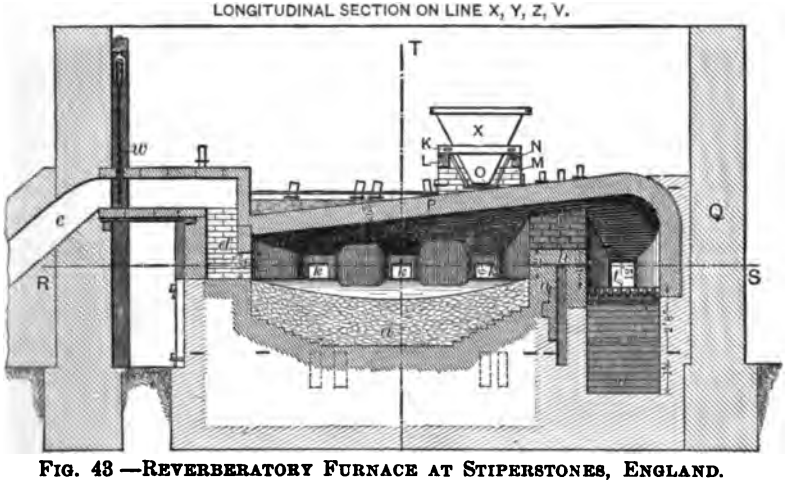


FIG. 43.—REVERBERATORY FURNACE AT STIPERSTONES, ENGLAND.

The furnace is built of red brick and lined with a full course of firebrick, which is indicated by heavy shading. The foundation is 3 ft. deep; the space in the center between the bent ends (*j j'* in Fig. 44) of the lower tie-rods is filled with sand. The upper part of the foundation is solid and slopes from the back and both ends to the front. The edges of the last course of brick are left sharp, as shown by the zigzag line in Figs. 43 and 44. In determining the slope, a line 3 ft. 6 in. long is drawn from the taphole, and the points α and β located at a distance of 1 ft. on each side of its inner extremity. To them, lines are then

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drawn from the corners γ and δ, starting at an elevation of 2 ft. 8 in. The side walls are raised slightly over the roof P, and have a total height of 5 ft. 8 in., measured from the floor. The space between the two is filled with sand R'.

The binding of the furnace is clearly shown in the figures. The buckstays are 6 ft. 6 in. long, the tie-rods being slipped over them and tightened with wedges. The upper tie-rods reach over the furnace; the lower ones, $j\,j'$, are turned down 1 ft. at a distance of 4 ft. 6 in. The castings are not shown with sufficient distinctness in the drawings.

The taphole plate (Fig. 44 and y in Fig. 45) is 5 ft. long and

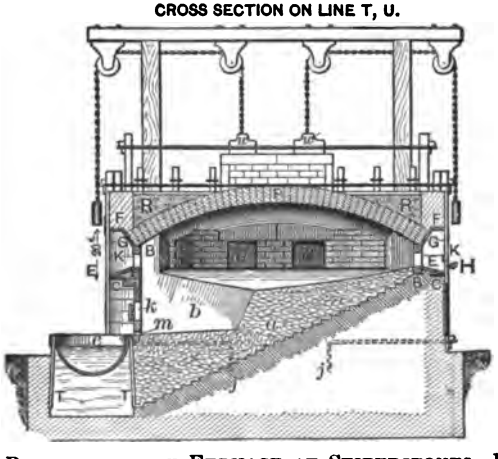


FIG. 44.—REVERBERATORY FURNACE AT STIPERSTONES, ENGLAND.

20 in. high; it is 2 in. thick at the bottom, increasing to 3 in., which it reaches at a height of 6 in. and retains for the rest of the distance. Eight inches above the center of the lower edge is the taphole m, and 4 in. above this begins a narrow vertical opening (4 by 9 in.), with a hinged door p only 8 in. high, so that when the door is closed a small current of cold air may pass over the molten lead in the well. The taphole plate is overlapped on each side by five cast-iron plates (4 ft. long and ½ in. thick), placed horizontally (z and z' in Fig. 45). They extend from the working opening down to the taphole.

The back-plate (5 ft. long, 20 in. high, and 1 in. thick) is placed just opposite the taphole plate. Its main object is to pro-

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tect the brick from the gray slag which is drawn out from the back door. To facilitate this, two hooks, *H* (Fig. 44), carrying an iron rod (3 ft. 6 in. long and 1 in. in diameter) as support for the working-tools are placed inside of neighboring buckstays.

The bridge-plate *n* (Fig. 43), counteracting the longitudinal thrust of the hearth, gives strength to the 4-in. air-space *o*, and prevents any leakage of lead. It is 6 ft. long and 20 in. high. From the upper edge, which is on a level with the grate bars, it is 3 in. thick for a distance of 6 in., then suddenly increases to

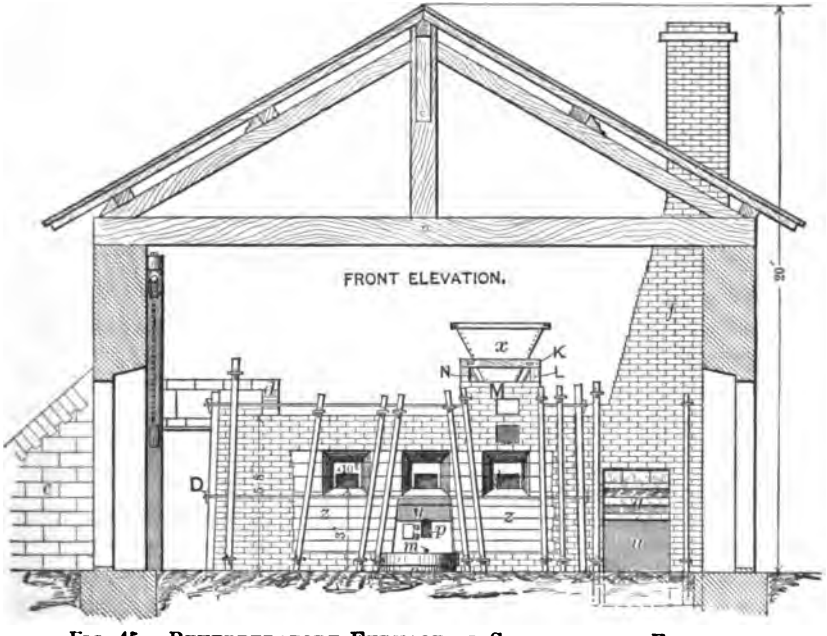


FIG. 45.—REVERBERATORY FURNACE AT STIPERSTONES, ENGLAND.

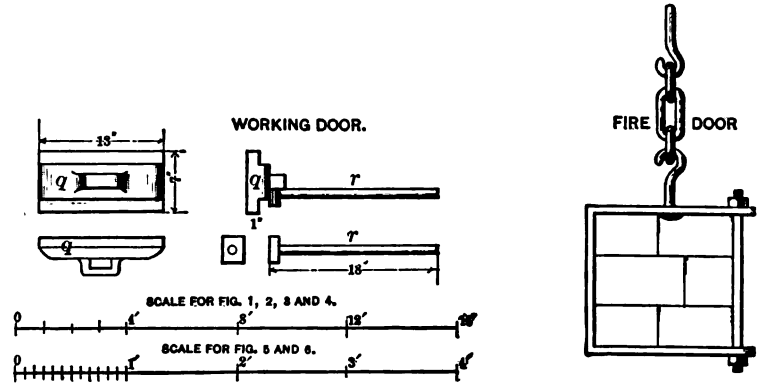
double that thickness for another 6 in., and for the last 8 in. returns to its original thickness.

The working openings *k* have a number of castings. The door-frames (*B*, Fig. 44), inclosing each an opening 10 by 6 in., are 4 in. square in cross-section. They are protected on the sides by firebrick, and on the bottom by the hearth. On the level of the grate-bars, or 2 ft. 8 in. above the floor, are two horizontal plates, *C*, 6 in. wide and ½ in. thick, that extend below the six working openings. On them rest two inclined (3 : 8)

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plates E, 8 in. wide and ½ in. thick, which abut against the bottom of the door-frames. Two plates F of the same size as C, form the upper part of the linings. The skewbacks G (9 in. wide and ½ in. thick), resting on these upper plates F, and the door-frames B, support the roof. The sides of the working openings are lined with ½-in. jamb-plates, as shown in Fig. 42. The cast-iron working doors g, with handle r (Fig. 46), are placed against the door-frames B.

The cast-iron kettle c, in front of the tap-hole is 2 ft. 6 in. in diameter and 20 in. deep. It is imbedded in clay, which is inclosed on three sides by masonry and on the fourth by the slag-bottom. Its rim, 8 in. above the floor, is surrounded by an iron



FIGS. 46 AND 47.—REVERBERATORY FURNACE AT STIPERSTONES, ENGLAND.

hoop, 4 in. wide and ½ in. thick. Wedges T are driven between it and the kettle to prevent any lead, that may penetrate through the joints and collect under the kettle, from raising it up.

The fire-opening t, 10 by 12 in., has an iron casing (Fig. 42). It is closed by a swinging fire-door (Fig. 47), consisting of fire-brick held together by a wrought-iron frame. The grate consists of two sets of cast-iron grate-bars of eight each, supported at the ends by cast-iron crosspieces.

The hopper (x, Figs. 43 and 45) is a truncated sheet-iron pyramid 36 and 6 in. square at the two ends and 3 ft. high. It is suspended in a wooden frame, K (3 in. square), which rests on two iron cross-beams, L, 4 by 6 in. These are supported by brickwork, 14 in. high, forming the continuation of the sidewalls

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of the furnace. Four iron rods *N* (Fig. 43), fastened in the wooden frame *K*, have at their lower ends a sheet-iron frame *O*, on which is placed a movable slide, which closes the discharge of the hopper. The 6-in. opening in the roof of the furnace, over which the mouth of the hopper is placed, is indicated on the hearth in Fig. 42. It is 1 ft. 9 in. distant from the firebridge, and 2 ft. 6 in. from the working opening nearest the bridge at the back of the furnace.

When the furnace is finished the working bottom, made of coarsely broken gray slag and sand, is put in. Part of the material is spread out on the brick bottom, which has previously been made red-hot. It is heated till it becomes pasty, and is then patted with paddle and rake. A second part is made to adhere to the first, and so on until the entire bottom has been built up with successive layers of gray slag and sand, and has attained the desired form and thickness. The upper edge of the well is 10 in. below the working opening. The thickness of the working bottom increases toward the taphole. It is 4 in. at the back doors, 12 at the bridge, 14 at the flues, 30 at the front, and 8 at the back of the well.

The furnace, being built solidly, lasts a long time; the roof of the fireplace requires repairing every two years, that of the hearth every five years. The hearth is repaired after every charge if necessary.

This furnace has the representative form of the English reverberatory. Slight variations exist as regards detail. For instance, the foundation, instead of being built up solid at the back, sometimes has an arched vault* at *j* (Fig. 44), extending longitudinally, communicating with the air-space in the firebridge, and being accessible at the flue end of the furnace. Another variation is when the entire hearth is built on cast-iron plates† resting on rails supported by a brick pillar. Sometimes the general form of the hearth differs by having a gentle slope toward the taphole. This makes the sides of the well less steep than those of the drawings.

The tools used in the furnace are shown in Figs. 48 to 61. They are of wrought-iron, with the exception of the mold (Fig. 60) and the handles of shovel (Fig. 52) and ash-pit hoe (Fig. 55).

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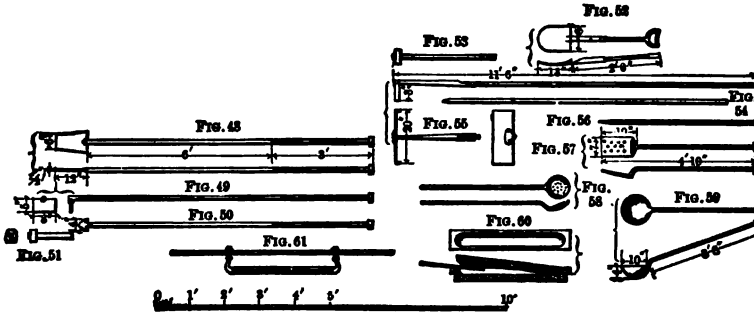
* Percy, "Lead," pp. 222-229. † Phillips-Bauerman, 1891, *Op. cit.*, p. 640.

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Figs. 48 to 51: Paddle, rake, old paddle used as chisel, and hammer to remove adhering slag are the tools for working on the hearth. Figs. 52 to 55: Coal-shovel, hammer to break coal, fire-poker, and ash-pit hoe are required for firing. Figs. 56 to 61: Tapping-bar, rectangular skimmer, circular skimmer, ladle, mold, and lead-carrier are used in handling the lead.

Method of Working.—To simplify the description the doors may be designated by numbers 1, 2, 3 in front, and 4, 5, 6 at the back, starting both times from the bridge.

The charge, 2,350 lb., is let fall from the hopper through the roof into the furnace, still red-hot from a previous charge, the dampers having been closed. The lead is left in the well and covered with lime. The ore is spread over the hearth through



FIGS. 48 TO 61.—TOOLS USED WITH THE REVERBERATORY FURNACE AT STIPERSTONES, ENGLAND.

doors 1, 4, and 5, with rakes. It decrepitates and gives off vapor of water. Then the dampers are slightly raised and the fire is gradually increased for an hour and a half. During the first two hours (first firing) the ore is turned over four times with paddles through doors 1 and 4, the other doors being kept closed. The paddling requires only a few minutes. When the first firing is nearly accomplished feeding of fuel is stopped, and doors 1, 2, 4, and 5, fire-door and dampers are thrown open and the charge cooled (first cooling). This lasts for half an hour. The grate is then freed from clinkers, and the charge, nearly 6 in. deep next to the bridge, which has fritted, is broken up and turned over. Any ore that had fallen into the well is raked up on the furnace-bed. The doors are now closed, the dampers lowered, the grate is well filled with coal, and the second firing begins. This lasts

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from 55 to 60 minutes, and is followed by a second cooling. Some lead now flows into the well. Near the bridge and toward the center of the furnace parts of the charge have begun to fuse, while at the flue the ore is only sintered. The charge is worked as in the first cooling, door 6 now being also used. After ten minutes the lead which has accumulated is tapped from the well into the kettle. The taphole is closed from the inside by inserting a clay plug fixed to a wooden handle through the little taphole door and pressing it in until the clay oozes out in front. Before tapping, fine coal and wood-shavings are put into the kettle to pole the lead. It is then stirred vigorously for six or seven minutes with the rectangular skimmer to bring all the impurities to the surface. These are removed, first with the shovel and then with the rectangular skimmer, and thrown back into the furnace, through doors 1 and 3, on both sides of the well. The charge, now freed from part of its lead, is turned over with paddles through doors 1 and 4. This ends the second cooling, four hours after charging. The doors are now closed, coal is put on the grate, and the dampers are lowered to begin the third firing. Meanwhile the lead in the outside kettle is skimmed and ladled into molds holding 120 lb. This takes about 20 minutes. During the third firing, which lasts two hours, the ore is turned over several times, care being taken to open the doors as little as possible. The furnace shows a bright-red heat when the third cooling begins. The charge is now worked with paddles for 15 or 20 minutes, and parts of it that have collected in the well are stiffened by the addition of lime and are raked on the hearth. The residue on the hearth is collected near the bridge and fine coal worked into it. The doors are again closed, the fire is urged for a quarter of an hour, and the residue turned over with the paddle through doors 1 and 4, and finally drawn out through door 5. Any repairing of the hearth that may be necessary takes place now, and the furnace is ready again for a new charge, seven hours from the time when the previous one was first introduced.

Two men work as partners in 12-hour shifts. Tabulated results are given in § 46.

The method of working near Holywell, North Wales, differs from that at Stiperstones. According to Percy* it is as follows:

After dropping the charge of 2,350 lb. on the hearth and

* "Lead," p. 232.

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spreading it over the upper part of the bed, the doors of the furnace and that of the fire-box are left open for an hour and a half to let the air have free access during the first roasting stage, while the damper is raised just enough for the gases to escape. Working doors 3 and 6 and the fire-door are now closed and the fire is urged. Lead soon appears. During this heating, which lasts two hours, doors 2 and 5 are closed, but 1 and 4 are kept open. Through these the charge is rabbled at intervals. Toward the end of this first reaction period lead begins to flow. Now doors 1 and 4 are closed, the damper is thrown open, the fire urged for 40 minutes, and the charge melted down. The furnace is then cooled for half an hour by throwing open all the doors. What charge remains on the hearth is rabbled, doors 3 and 6 are closed, slacked lime is thrown through door 2 on the charge, which has collected in the well, and worked into it through the taphole door. The stiffened residue is collected ("set up") near the bridge, as well as other parts that have been detached from the hearth. Doors 2 and 5 and the fire-door are now closed, the damper is lowered, and the temperature raised gradually for half an hour, when the damper is entirely thrown open, doors 1 and 4 are closed, and the fire is urged to melt down the residue, which takes 20 minutes. The fire-door and the working doors 1, 2, 4, and 5 are then thrown open, lime is added through door 2, and worked into the slag to thicken it. The lead is tapped, and the stiff gray slag raked out on the floor through door 5. The lead is poled as at Stiperstones, and the hearth repaired, if necessary. The entire time required to work the charge is six hours.

Two furnace-men and one helper work two charges in a 12-hour shift. Tabulated results are given in § 46.

The changes that take place in the ore during the process have been examined by Percy,* and are given here in their lead contents only.

Lead Compounds.	Hours After Charging.				Gray Slag.
	1½	3½	4½	4¾	
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
PbS	63.89	53.32	24.76	4.35	0.90
PbO	27.35	31.49	43.12	47.50	48.67
$PbSO_4$	3.82	4.78	6.94	14.03	9.85
Total Pb	88.16	78.66	66.22	47.86	52.88

* *Op. cit.*, p. 235.

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§ 45. The Silesian Method.—The characteristics of this method are a large charge, slow roasting, and a low temperature. It is not aimed to extract all the lead in the reverberatory, as this is supplemented by the blast furnace. The hearth is inclined toward the flue, beneath which the lead is collected and tapped at intervals into an outside kettle.

Lead-Smelting at Tarnowitz, Prussia.*—The ore is a mixture of sulphide and oxide lead minerals. Its composition was, in 1870, in round numbers:

	Per Cent.	Per Cent.
PbS	61	40
$PbSO_4$	11	9
$PbCO_3$	24	45
$(Ca, Mg, Fe, Zn)\,CO_3$	3	5
SiO_2	1	1 and less.

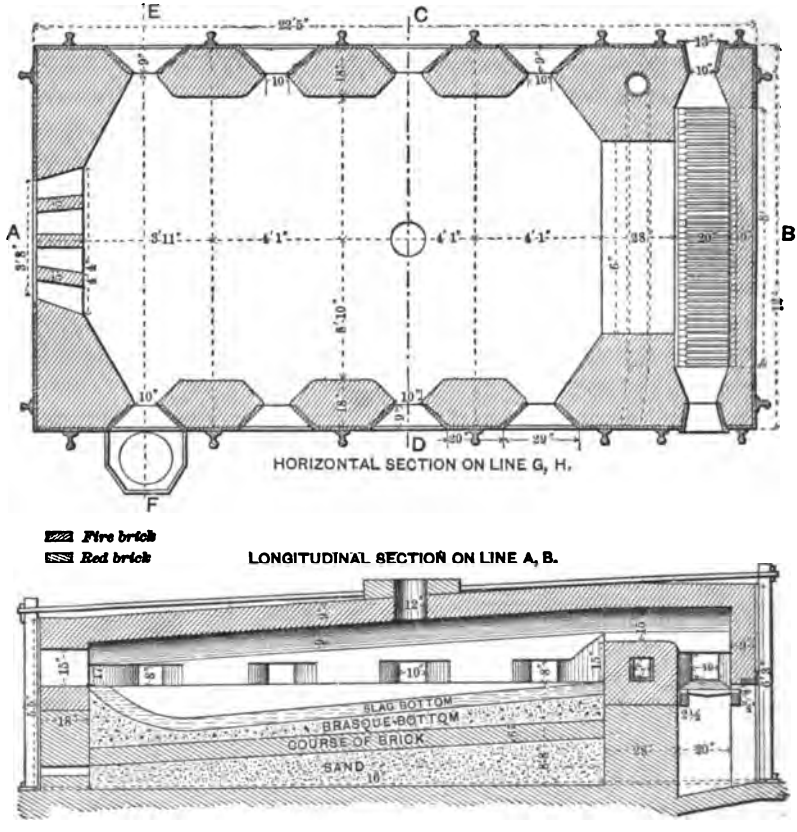
The furnace charges contained from 70 to 74% of lead and from 21 to 22 oz. silver per ton. At present the ore runs lower, the percentage of blende having increased considerably.

The construction of the latest furnaces is given in Figs. 62 to 67. After the detailed description of the reverberatory at Stiperstones attention need only be called to some of the principal features. The horizontal section (Fig. 62) shows the rectangular form of the furnace with four working openings on either side, the well being below the door nearest the flue end, the coolest part of the furnace. The grate bars are placed parallel with the axis of the furnace. The fire is stoked at the end and fed from both sides. Four branch flues lead the gases into one main flue. The bridge-wall has an air-passage. The entire furnace is incased in iron plates and bound by buckstays (iron rails) and tie-rods. In the longitudinal section, Fig. 63, the roof is seen to form a horizontal line from the fireplace to the beginning of the hearth, sloping thence in a straight line to the flues. The hearth, in Figs. 63 to 65, is built up differently from that of any other furnace. First a layer of sand is tamped down between the furnace walls so as to have the form of a trough inclining from bridge to flue. At the well the brick bottom, of red-brick

* *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, vol. xiv., p. 225, Teichmann; xix., p. 157, Wedding; xxxii., p. 94, Dobers and Dziegiecki; xxxiv., p. 292, Dobers and Althans; xli., p. 267, Saeger.

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set dry, replaces some of the sand. It forms a good support for the brasque bottom, which is followed by the slag bottom, consisting of tap-cinder melted down in the furnace. This covers the brasque bottom, wherever it would otherwise come in contact with working tools, leaving it exposed only at the well (Fig. 64).

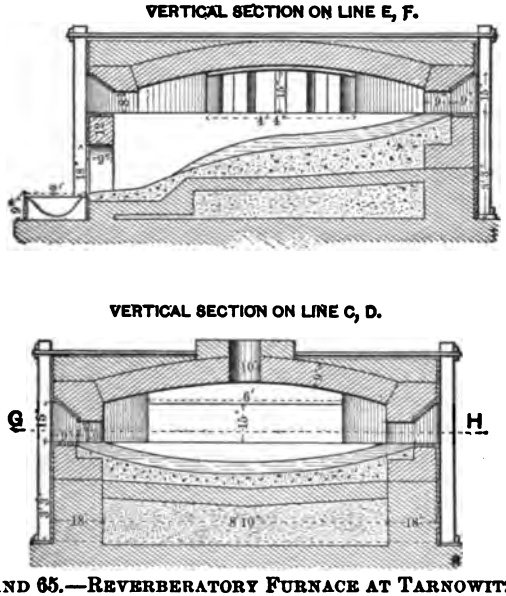


FIGS. 62 AND 63.—REVERBERATORY FURNACE AT TARNOWITZ, SILESIA.

During the run some residue adheres to this slag bottom and forms the smooth working bottom of the furnace. The hearth is seen in Fig. 65 to be trough-shaped from the bridge to the second working door; from there the front part (Fig. 64) slopes down to the level of the rim of the kettle, the lowest part of the well.

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The latest improvements of the Tarnowitz furnace, shown in Figs. 66 and 67, have been devised to make the work less dangerous and unhealthy. The gray slag, when drawn from the furnace, is not now discharged on to the floor or into a water-box, exposing the laborers to heat and fume or to explosions, but into a slag-pot *d*, placed in a niche *a*, where the slag may cool and the fumes pass back into the furnace through the opening. Further, the fumes issuing from the basin after the lead has been tapped into it do not pass into the building, but are drawn off by a telescope stack, *h*. The niche *a*, 30 in. wide, 26¼ in. deep, and



FIGS. 64 AND 65.—REVERBERATORY FURNACE AT TARNOWITZ, SILESIA.

26¼ in. high, forms an opening beneath the drawing door *b* (to receive the slag-pot *d*, which is brought on a separable car), the back and sides of which are inclosed by brickwork, the roof being formed by a heavy cast-iron plate which supports the brickwork and the hearth. The plate has an opening *c*, 7½ in. square (beginning 15 in. from the outside wall), through which the gray slag is raked into the pot. It is closed with an iron plate when not in use. When the slag is being drawn the front of the niche is closed by a movable sheet-iron door, having a peephole; all the fumes from the slag pass through the drawing

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opening *c* back into the furnace. When it has cooled sufficiently the slag-pot is covered with a piece of sheet iron and run out on the car. The telescope stack *h*, which is balanced by counter-weights, is lowered to the position *k* as soon as the lead is to be tapped, when the fumes will pass off through the stationary sheet-iron pipe *g*. The beneficial effect of these improvements is shown by the fact that while in the year 1887–88, 28.5% of the men became leaded, in 1891–92 only 0.8% were thus affected.

The tools required to work a furnace are four paddles, four large and two small rabbles, five shovels (two for lime, two for coal, and one for slag), three steel bars (two large ones and a small one), a tapping-bar, a skimmer, a ladle, a sample-ladle, two

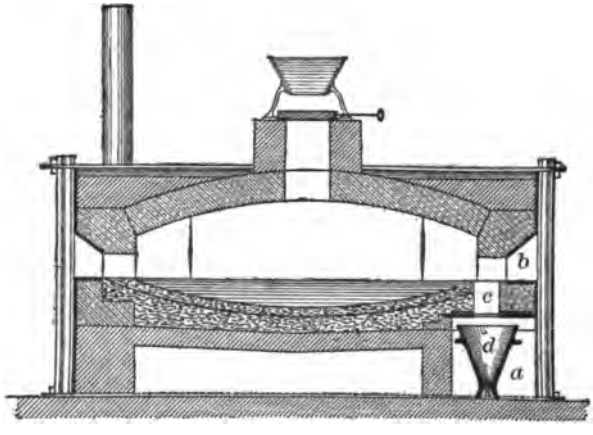


FIG. 66.—REVERBERATORY FURNACE AT TARNOWITZ, SILESIA.

slice-bars, a sledge, two hammers, and four hooks for handling the lead.

Method of Working.—The furnace, if new, is heated to a dark-red heat; the damper is then closed and the charge, crushed to pass a 5-mesh sieve, is let fall from the hopper through the opening in the roof and spread out evenly over the hearth by means of rabbles to a thickness of from 3 to 4 in. The fire is fed with cinders, and the temperature never allowed to exceed dark-redness, say 500° or 600° C. The galena decrepitates, the temperature rises, the ore becomes a dark-red, and after from three-quarters of an hour to an hour the roasting begins. During this time the charge is turned over once with the paddle. The

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working doors and fire-doors are opened, and the damper raised sufficiently to allow the sulphur dioxide and other gases to pass off. The ore roasts on the surface. When the fumes begin to lessen, samples are taken to see if a white crust of oxide and sulphate has been formed. This indicates that it is time to renew the surface, which is generally done by paddling or rabbling every 20 or 25 minutes, *i.e.*, about eight or nine times during the three or four hours required for roasting. Care is taken to prevent the ore from clotting.

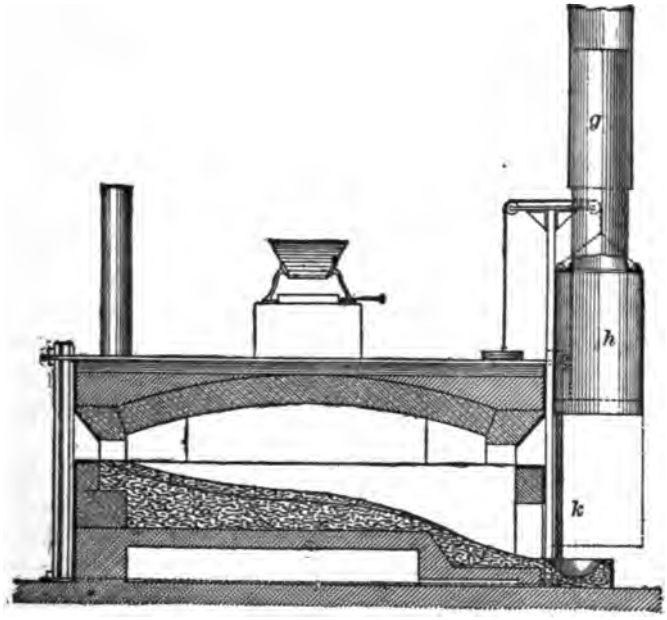


FIG. 67.—REVERBERATORY FURNACE AT TARNOWITZ, SILESIA.

Up to 1885 the roasting usually took about three or three and a half hours, but the gradual increase of blende in the ore brought the required time up to four hours. In 1886 it became necessary to make a change in working the charge. This could be done in two ways. Either the normal time could be prolonged or part of the ore roasted separately. The latter method was chosen, and now one-quarter of the charge, fine concentrates especially rich in blende, is roasted separately in a long-hearth reverberatory furnace and then added to the ore charge. Toward

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the end of the roasting period from 330 to 660 lb. of carbonate ore or fluedust, with 45% of lead, are added to increase the amount of oxides in the charge.

The grate is now cleaned, well filled with good coal, and the damper opened to raise the temperature. The reactions now set in. The roasted ore gradually softens, white fumes are given off, and lead begins to flow. Lime is added to prevent the lead from carrying down to the well particles of ore floating on top of it and to prevent liquefying of the charge. The lime is well worked in and the charge then turned over. The furnace becomes filled with fumes of lead, the damper is raised to remove these, and fusing of the charge is prevented by regulating the fire and the damper and by adding a few shovelfuls of lime at a time. At from an hour to an hour and a quarter after the reaction has set in (three hours with a new furnace) the well is full of lead. This is tapped into the outside kettle, after the telescope stack has been lowered. The dross floating on the surface is skimmed off and put back into the furnace and the impurities held in suspension by the lead are removed by stirring in slack coal (poling). This second dross is put aside and the lead ladled into molds. During the reaction stage the working doors have to be open on account of the necessary rabbling and paddling; thus air enters, cools the furnace, oxidizes the charge, and the flow of lead begins to cease.

At this stage drosses obtained from melting down base bullion in the desilverizing plant are added. They consist principally of a mixture of lead sulphide and lead, assaying 75% lead and 9 oz. silver per ton. At the same time 1,100 lb. of oxides low in silver, containing from 75 to 80% lead oxide and from 19 to 20% zinc oxide, are charged. They result from refining lead, desilverized by the Parkes process, by means of steam (§ 114).

The doors are then closed and fuel is piled up high on the grate to insure a smoky flame. This reduces lead sulphate to sulphide, and soon the flow of lead begins again. The doors are opened, the charge is worked, and its liquefying prevented as before. From two and a half to three and a quarter hours after the first reaction sets in the well is again filled with lead. This is tapped and lead and dross are treated as before. In this way all together five reactions are caused, the amount of lead obtained becoming gradually smaller and the charge more dry. Before

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the last reaction takes place the mixture of dross and fine coal, obtained from poling the different leads in the kettle, is thrown into the furnace. It is worked into the charge, which is then covered with fine coal. The temperature is raised and the last yield of lead obtained. The damper is now opened fully and the residue drawn from the back of the furnace through opening *c* (Fig. 66) into the slag-pot. Formerly it was drawn into a box filled with water, so as to prevent fumes of lead from passing into the building, and care had to be taken to avoid explosions. The hearth is repaired with a mixture of crushed residue and slacked lime, which is beaten down with the paddle and rammed into holes with a bent bar. When this is finished, a little lime is spread over the hearth and a new charge dropped from the hopper. The whole reaction period lasts seven hours.

If the hearth should become so incrusted as to raise the charge too high to be protected by the firebridge from the direct action of the flame, the temperature must be raised so as to soften the accumulated residue that it may be removed. A hearth lasts about three months.

The results from one year's (1865) work in a smaller furnace, 11 ft. 9 in. by 10 ft. 10 in., with six doors, and with ore low in blende and without the different additions, were: 10,000 lb. of ore (assaying 72.97% lead and 21.76 oz. silver) gave 6,384 lb. base bullion, assaying 32.95 oz. silver; 1,592 lb. residue (38.80% lead and 3.93 oz. silver); 275 lb. fluedust (50.00% lead and 0.02 oz. silver); showing that in the reverberatory 87.49% of all the lead and 99.9156% of all the silver was recovered in the form of base bullion. Of the 12.51% of lead, forming the difference, 9.31% were recovered in the blast furnace, so that the actual total loss amounted only to 3.2%.

The consumption of fuel was 4,600 lb., or 46% of the ore charged, and of lime 100 lb., or 1%. Additional data are given in the next paragraph.

The following analysis of residue was made by Pietsch in 1865: PbO, 24.375%; $PbSO_4$, 13.269%; $PbSiO_3$, 12.373%; ZnO, 22.857%; FeO, 8.957%; FeS, 1.823%; CaO, 11.190%; C, 4.821%; Al_2O_3; and MnO, traces; silver, 0.015% (=4.360 oz. per ton)—total, 99.680%.

The fluedust, in 1881–82, forming 2.91% of the ore charged,

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EXAMPLES OF SMELTING IN REVERBERATORY FURNACES.

Items.	Carinthian Method.					English Method.		Silesian Method.	
								Tarnowitz, Silesia.	
	Raibl, Carinthia.	Engis, Belgium.	Granby, Missouri.	South-west Missouri.	South-east Missouri.	Holywell, Flintshire.	Stiperstones, Shropshire.	Old Furnace.	New Furnace.
The Furnace.				*					
Hearth, length	11′	6′ 6¾″	9′	9	10′	12′	9′ 9″	11′ 9″	16′
Hearth, width at bridge	5′	4′ 3⅛″	3′ 2″	3′	2′ 6″	3′ 6″	4′	.....	6′
Hearth, width at middle	5′	4′ 4¼″	3′ 2″	3′	4′	9′ 6″	9′ 6″	10′ 10″	8′ 10″
Hearth, width at flue	1′	2′ 3½″	3′ 2″	3′	2′	4′	3′ 3″		4′ 4″
Hearth, thickness of bed	6″	1′ 5¾″–2′ 8¾″	6″	6″	.....	2″–10″–26″	4″–18″–30″		4″
Hearth, life of bed, weeks	4-5	6	.....	.....	.....	.....	.....		13
Hearth, inclination to side or flue	21″ F.	17″ F.	18″ F.	—F.	—F.	22″ S.	24″ S.		27½″ F.
Grate, length	7′	3′ 7¼″	5′ 6″	6′ 6″	3′	4′ 6″	4′ 6″		8′
Grate, width	1′ 3⅝″	1′ 6″	2′ 6″	2′ 6″	2′	2′ 6″	2′ 6″		1′ 8″
Grate, depth below top of bridge	1′ 2⅝″	3′ 7¼″	.....	6″	.....	2′ 9″	1′ 4″		3′
Bridge, width	5⅝″	1′ 3¾″	8″–30″	.....	21″	24″	24″		28″
Bridge, length	3′ 7¼″	3′ 11¼″	30″	30″	24″	42″	48″		72″
Bridge, height above hearth	4-5″	None.	.....	6″	.....	12″	12″		15″
Roof, height above bridge	9″	8⅝″	14″	.....	6″–8″	9″	19″ †		12″
Roof, height above hearth at flue	12″	2′ 3⅛″	14″	.....	6″–8″	16″	20″		17″
Flue (leading to chimney), size	1′ × 1′ 3¾″	8¾″ × 2′ 3½″	8″ × 12″	.....	12″ diam.	12″ × 10″ / 10″ × 10″	12″ × 12″ / 11″ × 12″		4 (15″ × 20″)
Chimney, inside size	1′ 8″ × 1′ 8″	1′ 3″ × 1′	.....	.....	12″ diam.	3′ × 3′ 6″	.....		.....
Chimney, height	20′	.....	.....	20′	15′	.....	.....		.....
Grate area, ratio to hearth area	1 : 6	1 : 5.25	1 : 2.0	1 : 1.66	1 : 5.4	1 : 8	1 : 6.6		1 : 10
Grate area, ratio to section of chimney	1 : 0.14	1 : 0.47	.....	.....	1 : 0.13	1 : 0.98	.....		.....

Similar to new furnace.

* Kemp, *School of Mines Quarterly*, ix., p. 212. † Height of roof above grate, 38″.

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EXAMPLES OF SMELTING IN REVERBERATORY FURNACES—*Continued.*

Items.	Carinthian Method.					English Method.		Silesian Method.	
								Tarnowitz, Silesia.	
	Raibl, Carinthia.	Engis, Belgium.	Granby, Missouri.	South-west Missouri.	South-east Missouri.	Holywell, Flintshire.	Stiperstones, Shropshire.	Old Furnace.	New Furnace.
Work Done.									
Charge, weight in lb	370–420	1,375	1,500	1,500	.....	2,350	2,350	4,400	8,250
Charge, assay, per cent. lead	70	76	.....	.....	.....	75–80	77.5	70–74	.....
Charge, thickness on hearth	1⅛″–1½″	About 3″	.....	.....	.....	1⅝″ ?	3⅛″–4½″	3″–4″	3″–4″
Operation, length in hours	10–12	12	12	9–11	.....	6	7	12	12
Men, number in 24 hours	1.5	4	2	2	.....	6	4	4	6
Gray slag, amount in lb	95	275	.....	.....	.....	288	.....	600	2,475
Gray slag, per cent. of charge	24	12	.....	.....	.....	12	.....	15	30
Gray slag, assay, per cent. lead	4	17–20	55	.....	.....	55	.....	38.8	56
Of lead charged, recovered per cent., in bars	93	92.5	.....	.....	.....	81	88	87.5	50
From gray slag and fluedust	.....	2.3	.....	.....	.....	8	.....	9.3	.....
Total loss in lead	7	5.2	....	.....	.....	11	12	3.2	.....
Outlay for Labor, Fuel and Material Per Ton of Ore of 2,000 lb. Avoirdupois.									
Labor, hours	90	61	36 aver.	20	.....	15	12	11	9
Fuel, charcoal, lb	.....	.....	.....	.....	.....	.....	.....	.....	.....
Fuel, bituminous coal, tons	.....	1.10	.....	.....	.....	0.57–0.76	0.56	0.46	.....
Fuel, wood, cords	1.60	.....	1.10 aver.	0.83	.....	.....	.....	.....	.....
Material, lime, lb	.....	.....	.....	.....	.....	.....	.....	20	.....
Reference	§ 43, *A.*	§ 43, *B.*	§ 43, *C.*	§ 43, *C.*		§ 44	§ 44	§ 45	§ 45

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had the following composition, according to Dobers and Dziegiecki:

	I.	II.	III.
	Per Cent.	Per Cent.	Per Cent.
PbO	66.44	66.53	65.79
ZnO	3.77	4.55	4.05
Fe_2O_3	0.50	0.30	0.35
SO_3	27.48	27.11	27.79
Insol.	2.14	1.20	2.30
Totals	100.33	99.69	100.28

§ 46. Comparison of Reverberatory Methods.—To facilitate comparison, the main data of the furnaces discussed have been brought together in the table on pages 112 and 113:

In comparing the amounts of ore treated in 12 hours at the different smelting works, the order in which they are placed in the table shows a steady increase from Raibl to Tarnowitz. The figure for Tarnowitz, 8,250 lb., requires the explanation that the charge contains a considerable proportion of oxidized ore, which shortens the time required for roasting. If the ore were pure galena, twice the time, or eight hours, should be allowed. This would make the amount for 12 hours 6,187 lb. The large amount of ore treated is due to the size of the furnace. As to the amount of labor required per ton of ore, the table shows that Tarnowitz uses less and Raibl more than any of the other smelting works. With fuel the same is the case. As the wear and tear of a furnace depend on the height of the temperature, the Carinthian and Silesian furnaces will outlast the English ones, other things being equal. If, finally, the amount of lead recovered is considered, it is clear that slow roasting yields more than quick roasting (Carinthian and Silesian *vs.* English), and that recovering in the reverberatory only that amount of lead obtainable at a low temperature and not melting the charge, gives a higher percentage of lead (Tarnowitz and Engis *vs.* Raibl and English methods). The inferences to be drawn are clear. They are stated by Cahen,* and Grüner† has formulated them at length. He says (Percy's translation):‡ "Rich, pure, non-quartzose ores ought always to be treated by this latter (the roasting and reaction) method. The operation ought to take

* *Op. cit.*, p. 117. † *Op. cit.*, reprint, p. 511. ‡ "Lead," p. 491.

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place in large reverberatory furnaces, with easy access of air, provided with a single fireplace and a receiving basin, internal or external, placed in the least heated region of the furnace. The operation ought always to be conducted slowly, and to consist of two phases very distinct, *roasting* and *reaction*. For roasting, the layer of ore must never exceed from 3.15 to 3.54 in. in thickness. Roasting is to be effected at a low temperature, and ought to proceed as far as the theoretical limit of one equivalent of sulphate, or two equivalents of oxide, for each equivalent of sulphide. After the first firing, which produces lead, and fresh roastings and firings twice or thrice repeated, the rich residues must be withdrawn from the reverberatory furnace, without having recourse to *ressuage* (reduction of rich residue in the same furnace immediately afterward), but rather by practicing this *ressuage* in a blast furnace."

Why in our own lead districts the reverberatory practice has not made more headway is to be answered in two ways. In the lead-silver districts the ores have on the whole not been of sufficient purity and richness to warrant the use of reverberatory methods. In the Mississippi Valley, where the ore is of the quality required for the process, the question of skilled labor has had some influence, but whether justly so is very doubtful.

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CHAPTER VII.

SMELTING IN THE ORE-HEARTH.

§ 47. Introductory Remarks.—The process carried on in the ore-hearth is mainly the roasting and reaction process. It resembles that in the reverberatory furnace, with this difference, that oxidation and reduction go on simultaneously, and that carbon in addition to sulphur acts as a reducing agent, the charge floating on a bath of lead. The lead oxide and sulphate, as soon as formed, react on undecomposed sulphide, some lead oxide is reduced by carbon, and the liberated lead trickles through the charge into the hearth bottom, overflowing into an outside kettle.

The same conditions are as necessary for the hearth treatment as for the reverberatory, with the exception that the ore should be coarser. The smallest permissible size is that of a pea, and nut size is desirable. If fine ore is to be treated, it must first be agglomerated in a reverberatory, as in the ore-hearth it would be blown away. It requires power and a blower. Then much lead is volatilized; hence it is not suited for argentiferous ores. The comparative loss at Raibl,* where ore-hearth and reverberatory worked on the same ore, was, in 1888, 10.6 to 7.3%. It requires purer and higher grade ore than the reverberatory, but consumes less fuel. According to Tunner,† a furnace similar to the one at Raibl consumed, per 100 lb. of galena, 7.31 cu. ft. of wood, while the ore-hearth required only 2.90 cu. ft. The ore-hearth has three times the capacity of the air furnace. The cost of treatment per ton of ore is the same with a single ore-hearth; with several running side by side it becomes less, as only one set of men is necessary to run the more powerful engine, and this con-

* *Oesterreichische Zeitschrift für Berg- und Hütten-Wesen*, 1888, p. 320.
† *Leobener Jahrbuch*, 1852, I., p. 262.

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sumes relatively less steam than the smaller one. That the ore-hearth cannot compete with the English or Silesian furnaces as regards capacity and cost is clear. It has, however, one advantage over all reverberatories, that is quickly started and stopped without much consumption of fuel or loss in heat, and thus serves its purpose in extracting at intervals from small amounts of non-argentiferous ore the major part of the lead. This is probably the reason why it found such favor in the Mississippi Valley, where small amounts of ore were often treated, and still are, by men who have mined it themselves.

The ore-hearth as worked at Joplin, Mo., is exceptional and will be treated further on (§ 52).

The practice in the different ore-hearths is so very much the same that nothing from a general point of view need be said about it.

The products are similar to those of the reverberatory; there is, however, an intermediary product, a mixture of ore, slag, and fuel, called browse in England, which goes back to the charge in the ore-hearth. The slag contains much lead, and is smelted in a small blast furnace, known as the slag-eye furnace.

§ 48. Influence of Foreign Matter.—It has already been said that the ore subjected to hearth treatment must be purer and richer than is necessary for the reverberatory furnace. This is because the foreign matter shows its bad influence in a more marked degree. The chemical action is, however, the same as that described in § 41.

§ 49.—*Description of Ore-Hearths.*—An ore-hearth, being a small low fireplace surrounded by three walls, with a tuyere at the back, cannot show much variety in construction or in manner of working. Three slightly different forms have been chosen by way of illustration: (1) The Scotch Ore-Hearth; (2) The American Water-Back Ore-Hearth; and (3) The Moffet Ore-Hearth.

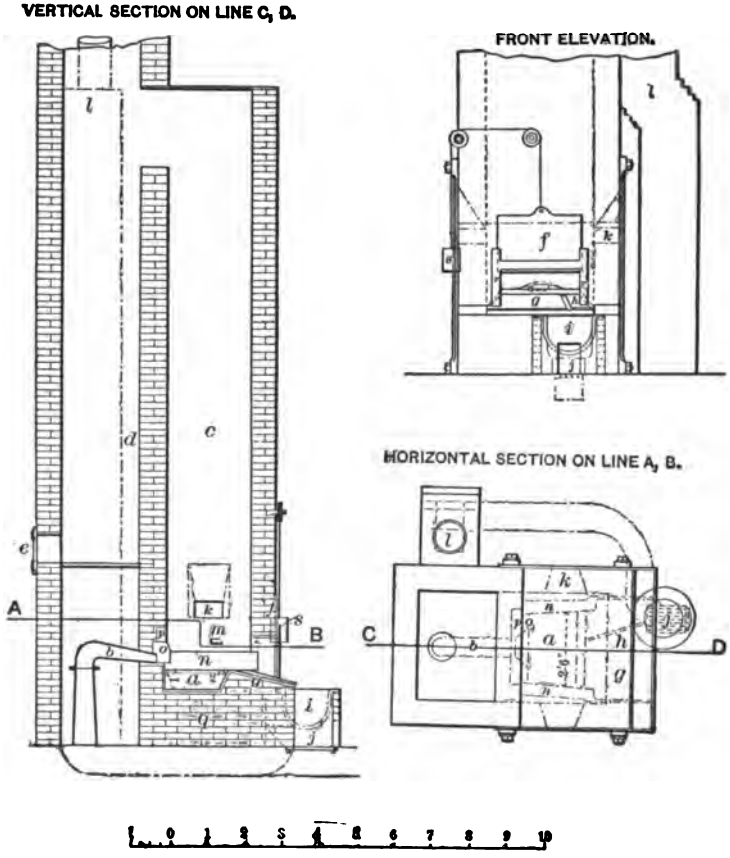
(1) *The Scotch Ore-Hearth.**—Figs.† 68 to 70 represent the furnace used by Messrs. Cookson & Co., near Newcastle, in the North of England. The cast-iron hearth-box or well *a*, which holds the lead, is set in brick-work *q*. It is 2 ft. from front to back and about 2 ft. 6 in. wide; it is 1 ft. deep and holds about

* Percy, "Lead," p. 278.

† "Eighth Annual Report of the Local Government Board," 1878-79, supplement containing the Report of the Medical Officer for 1878, London, 1879, p. 281.

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two tons of lead. In some furnaces the depth is only 6 in. and the capacity of the well about 1,340 lb. of lead. The work-stone *g*, an inclined plate, is cast in one piece with the hearth-box. It has a raised border on either side and at the lower edge, and a groove *h*, which leads the overflowing lead toward the kettle *i*,



FIGS. 68 TO 70.—SCOTCH ORE-HEARTH, NEWCASTLE, ENGLAND.

heated from a fireplace *j*, below, the gases passing off through a flue into the chimney *l*. On either side of the hearth-box and resting on it is a cast-iron block (bearer) *n*. Another cast-iron block *o* (back-stone, pipe-stone) is placed at the back. It is perforated for the passage of tuyere *b*, which enters the furnace

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about 2 in. above the surface of the lead in the well. Upon *o* rests the upper back-stone *p*, also of cast-iron. Thus lead and ore are entirely surrounded by cast iron. The fore-stone *m* appears to be rather small. The brick shaft *c* carries off the fumes; at the back is a "blind flue" or pit *d*, forming a sort of dust-chamber, to be cleaned through door *e*, the gases passing off upward. On the side of the shaft is the feed-door *k*, for the introduction of fuel in front of the tuyere and the removal of any slag adhering to it, the charge being fed from the front. The hearth has at the front a shutter *f*, sliding in a grooved

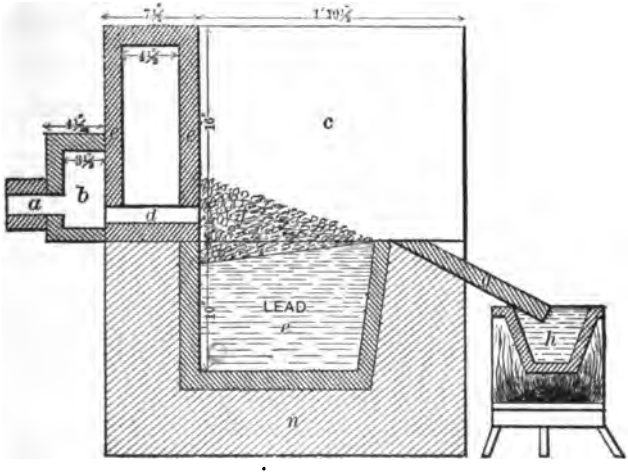


FIG. 71.—AMERICAN WATER-BACK ORE-HEARTH.

frame *r*; it is raised and lowered by means of counterpoise *s*. Peat, formerly used as fuel, is now replaced by bituminous coal.

It is characteristic for the method of working in the Scotch hearth that the process is non-continuous. After smelting from 12 to 15 hours the hearth becomes too hot and must be allowed to cool for about five hours before work can be started again.

(2) *The American Water-Back Ore-Hearth.**—Figs.† 71 and 72 show the larger-sized furnace with three tuyere-holes *m* at the back, where the older, smaller form had only one. The hearth-box *e* (filled with lead, the charge of ore and charcoal floating on

* Williams, "Industrial Report, Geological Survey of Missouri," 1877, p. 36; "Transactions of American Institute of Mining Engineers," v., p. 324.

† Broadhead, "Geological Survey of Missouri," 1873-74, p. 492.

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top) is here also set in brickwork *n*. It holds about 2,500 lb. of lead. The work-stone *g*, leading to the kettle *h*, forms a separate casting from the hearth-box. The three sides of the furnace are formed by a water-cooled cast-iron jacket, *c c*, 1½ in. thick, called a tuyere-plate, resting on the hearth-box. The water enters at *i*, and passes out at *k*. At the back of the tuyere-plate is the wind-box *b*. The blast enters this at *a*, and passes through three wrought-iron tuyere nozzles *d* (from 1 to 1½ in. in diameter) in tuyere holes *m*, into the hearth, at from 1 to 3 in. above the level of the lead. The hood placed over the furnace to carry off the fumes and gases is not shown in the drawings.

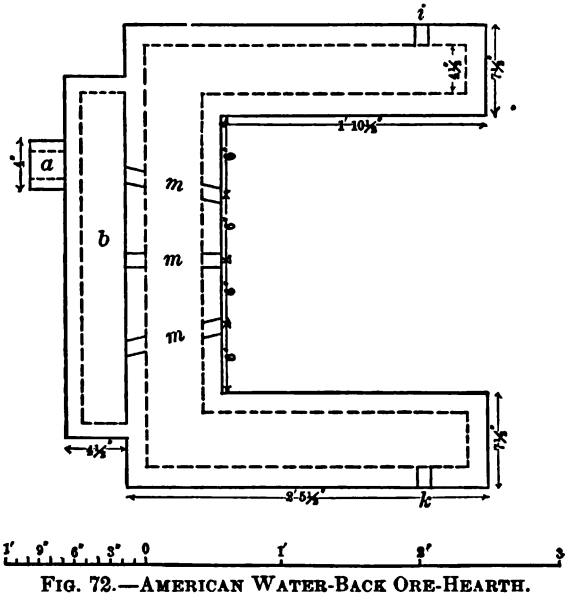


FIG. 72.—AMERICAN WATER-BACK ORE-HEARTH.

The work in the American ore-hearth is continuous, as distinguished from the Scotch hearth. This is made possible by water-cooling the sides of the furnace which protects the castings and the tuyeres. The fuel used is wood, charcoal, and bituminous coal.

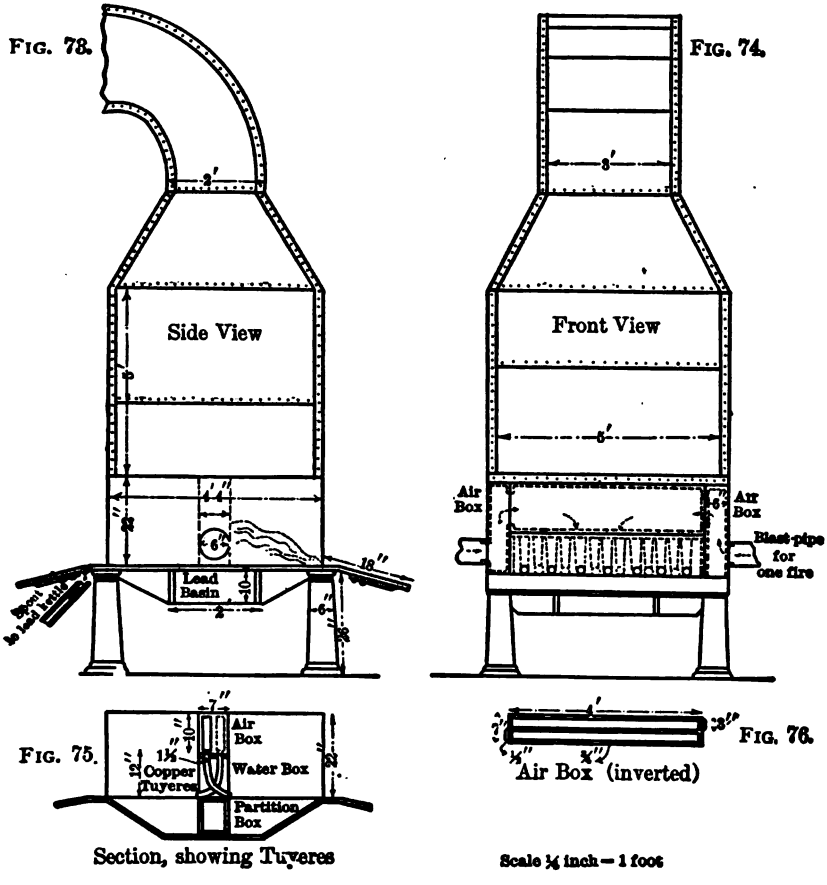
In the ore-hearth used formerly at Rossie,* N. Y., and for some time, also, in the Mississippi Valley, but entirely abolished now, this cooling was effected by letting the blast circulate instead of

* Percy, "Lead," p. 289.

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the water. The resulting hot blast caused much volatilization of lead.

(3) *The Moffet Ore-Hearth** (Figs. 73 to 76).—In Figs. 73 and 74 the entire furnace is seen to rest on four pillars. The



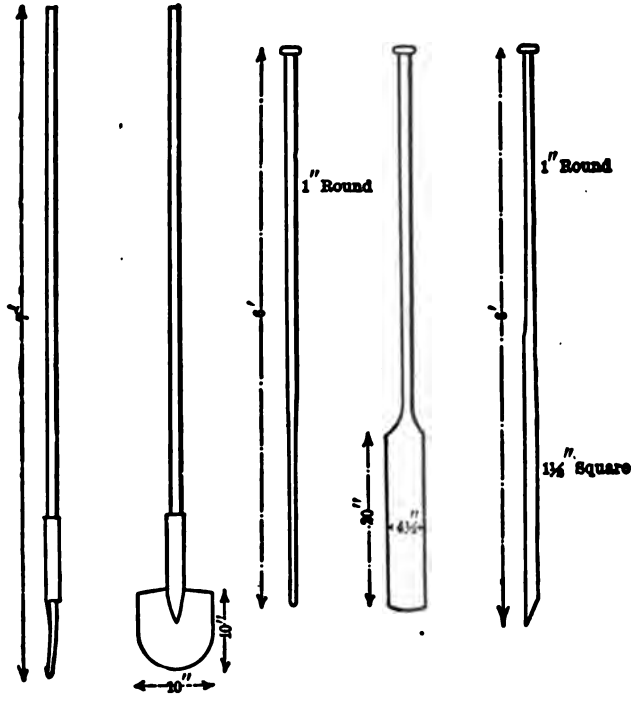
Figs. 73, 74, 75, and 76.—Moffet Ore-Hearth.

hearth-box (lead basin) is not set in brickwork, and the lead is thus kept cool. Two furnaces are set back to back, the fumes passing off under one hood. The lead runs through a separate spout near the top of the work-stone into a cast-iron

* Dewey, "Transactions of American Institute of Mining Engineers," xviii., p. 674; Clerc, *Engineering and Mining Journal*, July 4, 1885; Ramsay, *Scientific American Supplement*, May 14, 1887, No. 598; Holibaugh, "Lead and Zinc in Missouri," New York, 1895, p. 37.

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kettle 31 in. in diameter and 44 in. deep, not shown in the drawings. It is set in a cast-iron cylinder and heated from below, one cord of wood a week being required for the purpose. The two furnaces work independently. On the bottom of the lead basin rests the partition-box, having an opening near the bottom for the lead to enter. It serves as a support for the water-box, which cools the hottest part of the furnace, and upon which rests the air-box, consisting of two separate chambers (Fig. 76), where



FIGS. 77 TO 81.—TOOLS USED WITH THE MOFFET ORE-HEARTH.

the heated blast passes down through the water-box by means of fourteen 1-in. copper tuyere pipes, seven on either side. The working opening is 15 in. high. A No. 5 Baker blower furnishes the blast. The fuel used is bituminous coal. The reasons for the hot blast will be given in § 52.

The tools, a long-handled, round-pointed shovel, a round bar, a paddle, and a square bar are shown in Figs. 77 to 81. With

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the Scotch* ore-hearth the round-pointed shovel is used at some works for ore only, and a square-pointed one for working in the furnace; then a short scraper is introduced through the feed-door to remove slag adhering to the tuyere nozzle.

§ 50. Mode of Working in the Ore-Hearth.—A fire is first kindled with wood; then coal is added, and the blast started. This will soon set the fuel in a blaze; more coal is then added, ashes and clinkers are removed, and in a short time a body of glowing fuel is obtained, filling the entire hearth. Some residue from a previous run is then spread over the back part of the fire and the first charge, from 10 to 20 lb. of ore, given. This soon becomes red-hot, and the lead that is set free trickles through the body of the fuel and collects on the bottom. More ore is added. The contents of the hearth are then lifted up with the bar and kept open in order that the heat may be distributed through the entire mass. Parts that have become melted and form lumps are drawn out with the shovel on to the work-stone, and the slag is separated from rich residue, which is returned to the furnace. Ore and fuel are again added, and the operation continued until the hearth is full of lead.

The hearth is now in normal working order; it is filled with lead, and glowing fuel, mixed with partly fused and partly reduced ore, floats on top of it. From 12 to 30 lb. of ore are mixed with 1½ or 2% of lime, spread over the glowing floating mass, and then covered with a little fuel. It is now exposed for from three to five minutes to the action of heat and blast. After this, one of the furnace-men inserts the bar at different places into the lead, loosens and stirs the charge, and raises it slowly, for which purpose the paddle is sometimes used. The other man draws the semi-fused mass out from below with the shovel upon the work-stone; this allows the ore on the surface to sink down. What has been drawn upon the work-stone is broken up and the slag separated from the half-decomposed ore. The former is thrown aside (sometimes into a water-box); the latter goes back to the furnace, slacked lime being spread over it, if necessary. Any slag adhering to the tuyeres is then removed; some fuel is distributed in front of them and over the charge; fresh ore is spread on the fuel, and this again covered with fine fuel; then all is ready for a second operation. As the smelting proceeds

* Percy, "Lead," p. 282.

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EXAMPLES OF SMELTING IN ORE HEARTHS.

Name.	Furnace.					Ore in 24 Hours, lb.	Products.			Men in 24 Hours.	Material Consumed.					Reference.
	Front to Back.	Width.	Depth.	Sides.	No. Tuyères.		Lead in 24 Hours, lb.	Gray Slag in 24 Hours, lb.	Per Cent. of Lead Direct from Ore.		Bituminous Coal, Pounds.	Peat, Bushels.	Charcoal, Bushels.	Wood, Cords.	Lime, Bushels.	
Keld Head Mining Co., England...	28″	21″	12″	Cast iron.....	1	10,752	8,064	602	74.44	6	374	12	.....	....	.....	(a)
Leadhills & Warlock, Scotland....	22″	30″	6″	Cast iron..:.....	1	.....	6,720	.....	.....	6	500	....	.....	....	.....	(b)
Rossie Works, New York.....	24″	24″	12″	Air jacket	1	9,988	7,389	.....	73.98*	4	.....	....	.....	¾	.....	(c)
Granby, Missouri.....	22½″	24″	11″	Water jacket.....	3	9,000	7,500	.....	83.90	6	.....	....	27.6	....	2.5	(d)
Hopewell, Missouri.....	.....	20″	12″	Water jacket.....	1	6,000	1,464	.....	73.20	6	.....	....	20.0?	....	.....	(e)
Lone Elm, Missouri.....	22″	48″	10″	Water and air jacket....	7	27,000	12,032	5,247	45.00	6	2,160	....	.....	....	5.40 lb.	(f)

(a) Percy "Lead," pp. 278-283. (b) Sexton, *Engineering and Mining Journal*, Feb. 23, 1895. (c) Kerl, "Metallhüttenkunde," p. 30.

(d) Williams, "Industrial Report," p. 63. (e) Ibid., p. 65.

(f) Dewey, "Transactions of American Institute of Mining and Engineers," xviii., p. 674.

To the above statements of labor and fuel have to be added the engine-men and the fuel consumed under the boiler, which is one cord of wood for the three-tuyere Granby ore hearth ("Broadhead's Report," p. 494).

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the bulk of the lead that is set free trickles through the charge into the hearth-box and overflows through the groove in the work-stone into the kettle. Here it is sometimes poled before being ladled, syphoned, or drawn off through a spout into molds. Some lead passes off with the fumes and the rest goes into the slag.

Pattinson* calls attention to the following points in managing the ore-hearth. The amount of blast and its distribution through the entire charge should be carefully regulated, the half-reduced ore should be exposed on the work-stone to the oxidizing action of the air, and the additions of lime and fuel judiciously made.

Two men work as partners in eight-hour shifts. The results obtained with different furnaces are given in the table on opposite page.

The composition of some Missouri leads smelted in the ore-hearth is as follows:

	Granby. (*a*)	Hopewell. (*b*)	Lone Elm. (*c*)
	Per Cent.	Per Cent.	Per Cent.
As	0.00124	0.00583	0.00011
Sb	0.01065	0.00803	0.00146
Ag	0.00057	0.00219	0.00056
Cu	0.00780	0.00585	0.01782
Ni	0.00087	Trace.	0.00077
Co	.....	.....	0.00005
Fe	0.00867	0.00145	0.00686
Zn	Trace.	0.00156	0.00033
S	Trace.	.....	.....
Pb, by difference	99.96905	99.97509	99.97204
	100.00000	100.00000	100.00000

(*a*) Williams' "Industrial Report," p. 63. (*b*) "Transactions of American Institute of Mining Engineers." v., p. 326. (*c*) *Ibid.*, xviii., p. 687.

§ 51. TREATMENT OF SLAGS.†—The gray slags obtained in smelting lead ores in the ore-hearth consist of lumps of a more or less scorified mixture of gangue and various lead (zinc) compounds, which contains mechanically inclosed lead and often particles of fuel. The mechanical analysis of Lone Elm slag‡ gave 21.45% metallic lead and the remaining 78.55% of pulp showed the following composition:

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* Percy, "Lead," p. 286.

† Bergen, in "Commissioner Raymond's Report," 1875, p. 424.

‡ "Transactions of American Institute of Mining Engineers," xviii., p. 685.

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	Per Cent.	Per Cent.
Residue.		
SiO_2	1.97	
$PbSO_4$	0.24	
Fe_2O_3	1.67	
Al_2O_3	0.21	
ZnO	0.57	
		4.66
$PbSO_4$		4.94
Acetic Acid Solution.		
SiO_2	10.73	
PbO	38.55	
Fe_2O_3	1.23	
Al_2O_3	0.57	
ZnO	13.96	
CaO	11.49	
MgO	0.12	
		71.65
Nitric Acid Solution.		
PbS	14.73	
FeS_2	0.67	
ZnS	3.64	
		19.04
		100.29

The gray slag produced in the neighborhood of Leadhills and Warlock, Scotland, has, according to Sexton,* the following composition: PbS, 5.63%; $PbSO_4$, 10.36%; PbO, 34.88%; CaO, 10.00%; ZnO, 0.95%; $Al_2O_3+Fe_2O_3$, 18.20%; SiO_2, 20.00%.

The gray slag is smelted in a low rectangular blast furnace, usually 4 ft. in height, having an external crucible (the cast-iron lead-pot) and one tuyere at the back. The bed-plate, sloping from back to front, carries the firebrick lining of the furnace, and overlaps the lead-pot. A brasque bottom, consisting of equal volumes of clay and coke, is tamped on it, 5 in. at the back narrowing down to 1 in. at the front. The front of the furnace has a cast-iron plate with an opening near the bottom. This is closed by ramming in a breast of clay over a wooden plug. The lead-pot is divided into two unequal parts by a partition descending nearly to the bottom. The melted charge, "black slag," flowing through the opening made by the wooden plug, passes over the larger division, filled with charcoal, into a tank, through which a slow stream of water flows. The lead filters through the charcoal, collects at the bottom, and is removed at intervals from the smaller division. A hood serves to carry off the fumes. The dimensions of the furnace are:

* *Engineering and Mining Journal*, Feb. 23, 1895.

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	Inches.
Width of hearth	26
Depth, front to back	36
Height	46
Diameter of tuyere	3
Height of tuyere above bed-plate	10
Height of port	9
Width of port	12
Prolonged axis of tuyere strikes front lining of furnace above port	2
Slope of bed-plate to 1 foot	1½
Size of bed-plate	40×48
Size of front plate	30×36
Long diameter of lead-pot	36
Front to back of lead-pot	12
Depth of lead-pot	12
Thickness of lining, back and sides	9
Thickness of lining, front	5

The starting of the furnace is very simple. A wood fire is kindled on the bottom, the furnace filled with clean charcoal, and some blast let on. When the charcoal is thoroughly ignited a 6-in. bed of coke is placed on top and full blast (9 oz.) given. Clean slag of moderate fineness is then charged with the necessary coke, four shovels of slag to one of coke. To save the walls of the furnace from being eaten out too quickly, the slag is placed rather around the back and the sides of the furnace, and the coke in the front and in the center. The wooden plug in the clay breast is withdrawn half or three-quarters of an hour after the full blast is given, when the "black slag" that had collected will begin to flow. During the run the charcoal in the lead-pot is removed every two or three hours. After smelting about 16 hours with a slightly acid slag the lining has become corroded, and infusible masses have accumulated on the bottom. The charging is then stopped and the furnace blown out by running off all the slag, tearing out the breast, drawing the rest of the charge, and chilling with water. After allowing it to cool for eight hours, adhering crusts are removed with steel bars, the furnace is patched, and it is then ready for another run.

From 15 to 18½ tons of slag, assaying from 35 to 40% lead, are smelted in 16 hours, giving 7,500 lb. of hard lead.

Two furnace-men, four helpers, and one roustabout form the crew; the fuel consumed is two tons of coke and 22 bushels of charcoal, besides that for the boiler.

A more modern slag-eye furnace, shown in Figs. 82 to 84, is the one used at the Lone Elm works. It is in many respects similar to the one just described. The furnace has 11 tuyeres of

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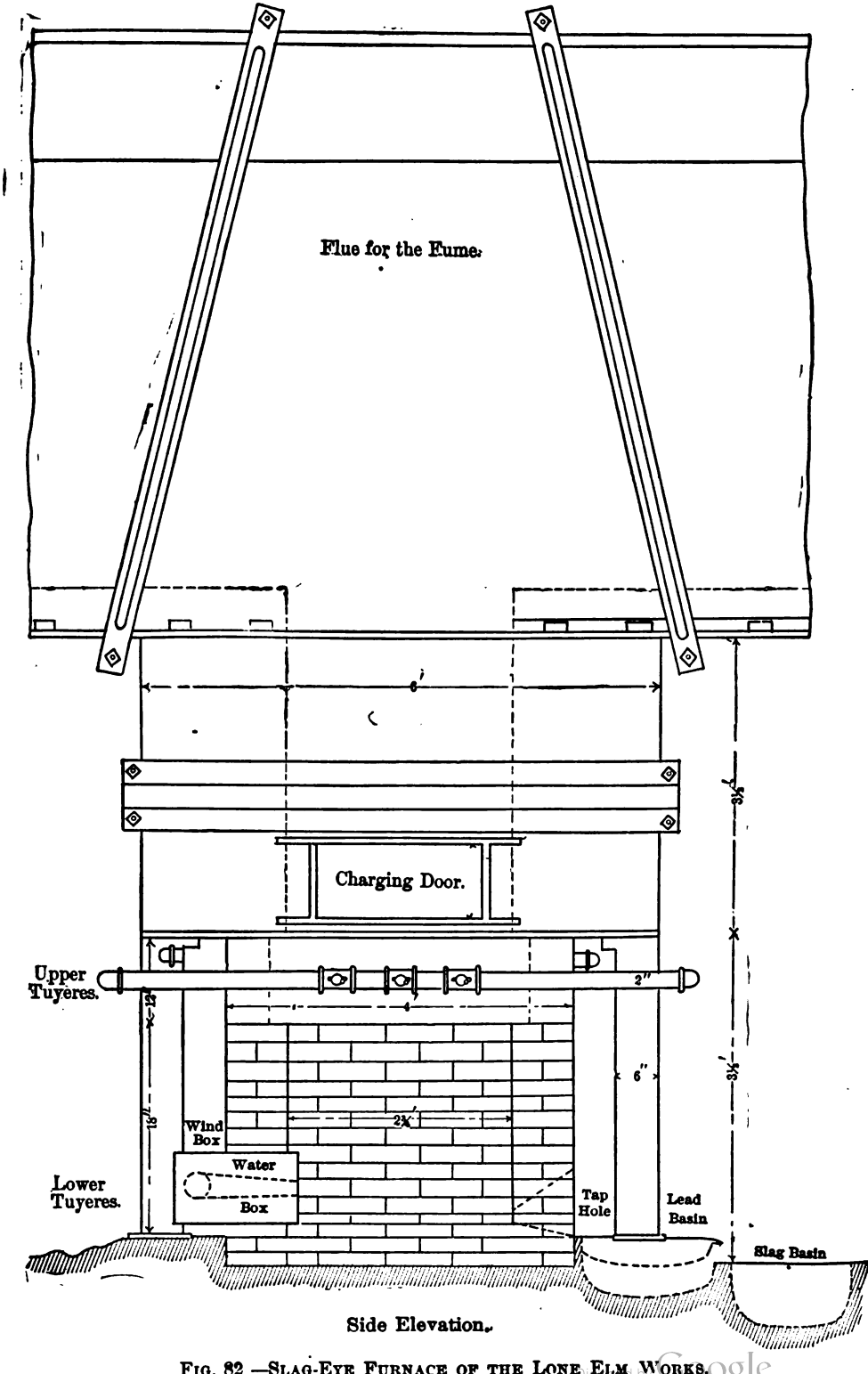
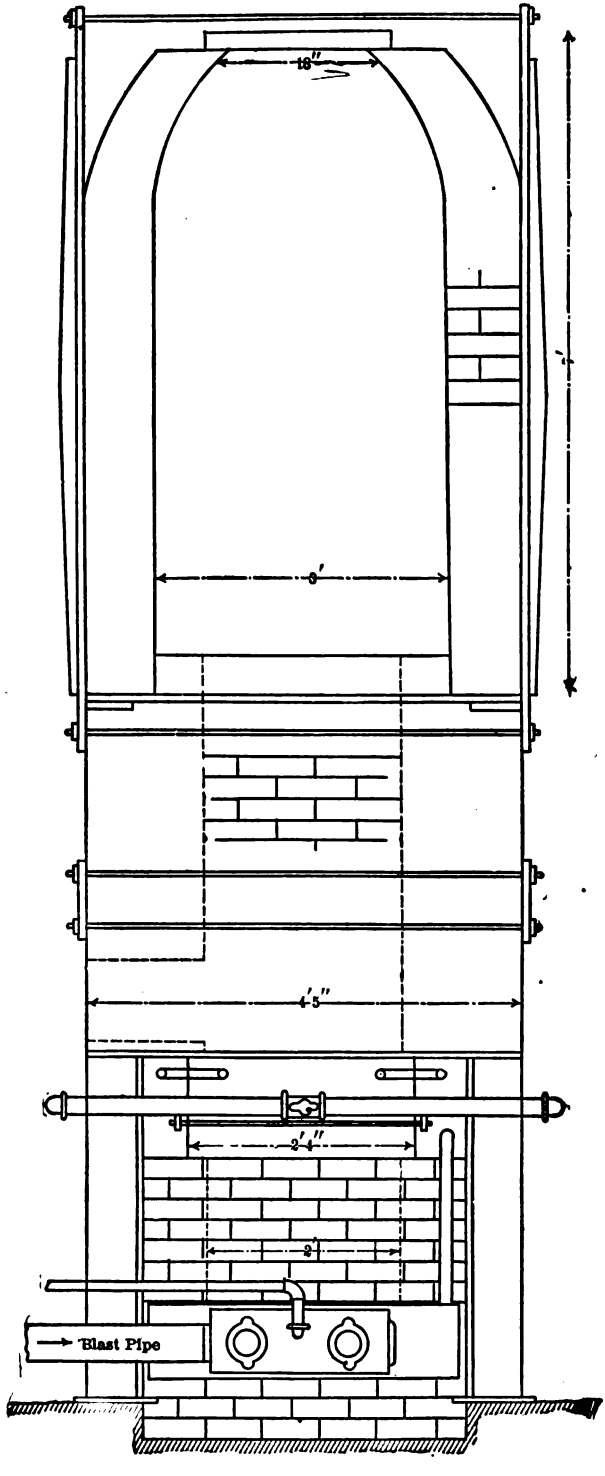


FIG. 82.—SLAG-EYE FURNACE OF THE LONE ELM WORKS.

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Elevation of the back.

FIG. 83.—SLAG-EYE FURNACE OF THE LONE ELM WORKS.

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1½ in. diameter, passing through water-boxes; seven of these are below the charging-door and four above the bottom, the blast being heated. (For the reason of this see next paragraph). The tools used are three 6-ft. bars, three shorter ones, two shovels, and a ladle. A furnace runs from 15 to 20 days without repairing. In the summer of 1891 the brick walls were replaced by water-jackets, which greatly prolong the life of the furnace. According to Petraeus,* with the brick furnace and little flux, the black slag sometimes ran as high as 10% lead, but the

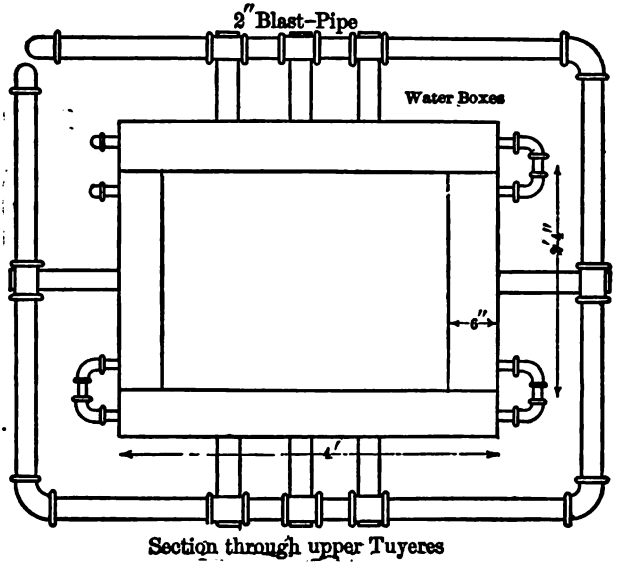


FIG. 84.—SLAG-EYE FURNACE AT THE LONE ELM WORKS.

general average was 5 or 6%. Since adopting the water jackets the slags run as low as 1.25% lead, but they average 4%. The reason for this is that in running the melted mass continuously from the furnace into an external crucible some lead remains entangled in the slag and is oxidized and scorified. The black slag commonly made contains SiO_2 27.5%, FeO 33.5%, CaO 22.0%, ZnO 8.5%. The percentage of lime varies greatly because the lime is added to the ore-hearth charge as it may be needed which makes the composition of the gray slag a fluctuating one.

* Private communication, October, 1895.

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If the black slag runs over 22% lime, the percentage of iron flux is decreased, with the unavoidable result that the percentage of the lead creeps up.

§ 52. Recovery of Fluedust by the Lewis and Bartlett Bag Process.*—One of the principal disadvantages of the hearth treatment is the loss in lead by volatilization. In some instances† condensing apparatus are used to save the fluedust. This subject will be discussed in connection with the smelting of argentiferous lead ores in the blast furnace (§ 93). With most American ore-hearths the fumes are allowed to go to waste. The Lone Elm works began as early as 1876 to collect and utilize them for pigments. This has been carried so far that in smelting lead ores the aim is not to produce as much metallic lead as possible, but to obtain large amounts of fume, which are condensed, purified, and sold in the market. This explains the use of hot blast in both furnaces and the position of the seven upper tuyeres below the charging-door in the slag-eye furnace. This method of working permits the treatment of large amounts of ore and shows also why only 50% of the ore charged is converted into metallic lead in the ore-hearth.

The conversion of the lead fumes into a marketable pigment comprises two operations:

(1) Cooling and collecting the dark ore-hearth fume, "blue powder," in the first bag-house, the "blue-room."

(2) Refining the blue powder in the slag-eye furnace, which is followed by cooling and collecting the resulting "white paint" in the second bag-room, the "paint-house."

The plan of the Lone Elm works and the details of plant are shown in Figs. 85 to 89.

The fumes from the ore-hearth are drawn off by a suction-fan, 6 ft. in diameter and 3 ft. wide, which makes 290 revolutions per minute. They pass through a water-jacketed brick flue into a brick dust-chamber (40 ft. long, 19 ft. high, and 6½ ft. wide, with a door on one side), where any coarse-grained particles of more or less changed ore and fuel are collected. They then pass out of the top of the chamber through a horizontal sheet-iron

* Dewey, "Transactions of American Institute of Mining Engineers," xviii., p. 674; Clerc, *Engineering and Mining Journal*, July 4, 1885; Ramsay, *Scientific American Supplement*, May 14, 1887, No. 593.

† Roesing, *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xxxvi., p. 1[illegible] (Lead Smelting in England); "English Government Report," quoted in § 49.

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pipe, 5 ft. in diameter, resting on 20-ft. iron pillars, to the fan, and thence through a 4-ft pipe, resting on 12-ft. pillars, to the blue-room. The pipe is sufficiently long (350 ft.) for the gases to cool in their passage through it.

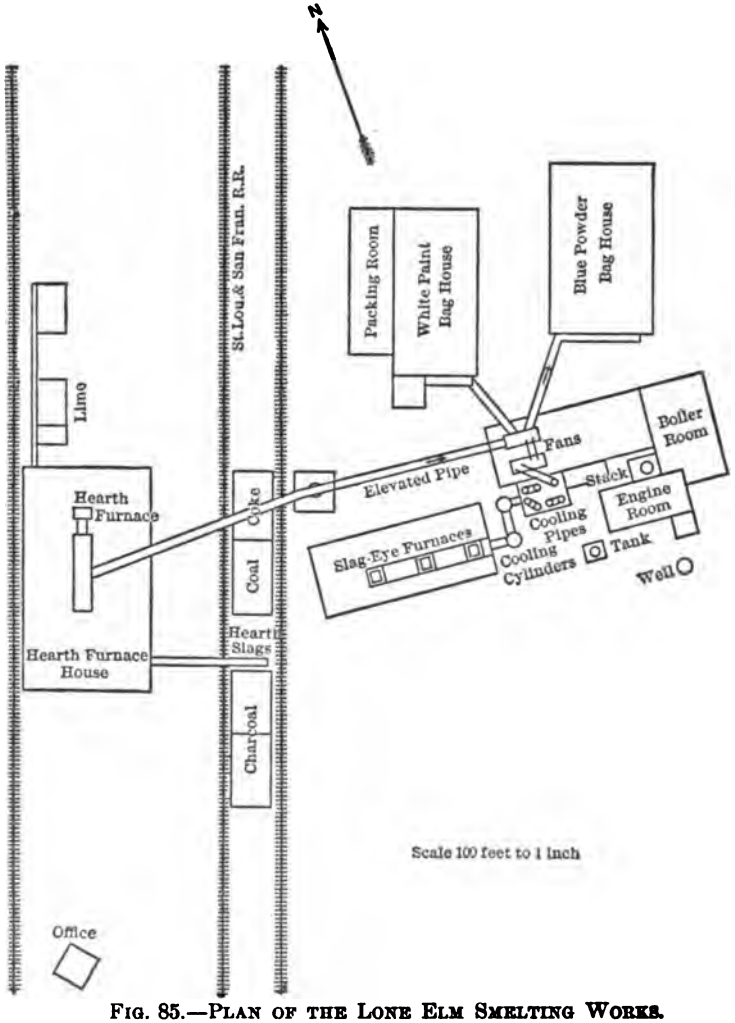


FIG. 85.—PLAN OF THE LONE ELM SMELTING WORKS.

The first bag-house is similar in construction to the second one, shown in cross-section in Fig. 89. It is a brick building

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(95 ft. long, 50 ft. wide, and 45 ft. high) divided into two compartments by a longitudinal wall, so that one may be shut off when it is necessary to gain access to the bags. Each compartment is divided into two stories, the lower being 12 ft. high. The divisions (columns, beams, etc.) are all made of iron pipe. In fact everything in the building is either of brick or iron, except the filtering bags. The lower story contains four rows of sheet-

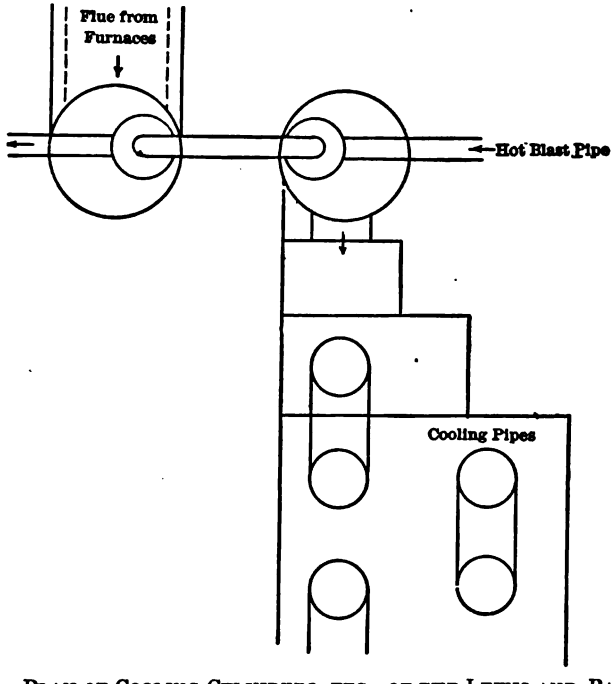


FIG. 86.—PLAN OF COOLING CYLINDERS, ETC., OF THE LEWIS AND BARTLETT BAG-PROCESS.

iron hoppers, extending the length of the building, which serve to collect the fume that has accumulated. They have the form of a truncated pyramid and are closed at their lower face by a sliding damper. They stand on four iron pipes, $3\frac{1}{2}$ ft. long, encased in refractory clay pipes. The upper face of a hopper is covered with sheet iron $\frac{1}{16}$ in. thick. This has 16 holes, 18 in. in diameter, from which thimbles, 12 in. high, project upward. Over these the lower ends of the bags, made of unwashed wool,

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60 in. in circumference and 33 ft. long (changing to 50 in. and 35 ft. when in use), are slipped and tied fast. The upper ends are tied with strong cord, with which they are suspended from beams near the roof. There are 800 bags in the bag-house, each costing $9. Between every two rows of bags is an iron scaffolding with iron footways placed at convenient heights, so as to make all parts of the building accessible.

The cooled gases, being pressed through the main pipe, enter four branch pipes, each of which passes through and connects a

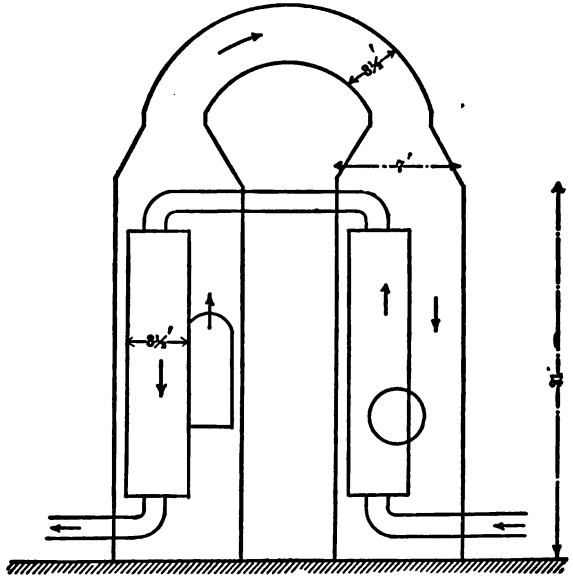


FIG. 87.—SECTION OF THE COOLING CYLINDERS SHOWING THE HEATING OF THE BLAST FOR THE SLAG-EYE FURNACE OF THE LEWIS AND BARTLETT BAG-PROCESS.

set of hoppers. The gases, laden with the dust, ascend into the hanging bags, where they are filtered, the fumes falling into the hoppers below. These are emptied once in two days, when the bags are also shaken to detach adhering fume. For this purpose the current of the gas is shut off, and men with aspirators pass quickly through the building, giving each bag a quick shake.

The collected fume is a very fine bluish-gray powder (blue powder), consisting mainly of lead oxide and sulphate, with some lead sulphide.

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BLUE POWDER.

	First Hopper, First Row.	Sixth Hopper, Second Row.	First Hopper, Fourth Row.	Tenth Hopper, Fourth Row.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
PbS	6.18	10.37	5.19	8.61
$PbSO_4$	45.84	43.55	46.88	43.57
PbO	44.44	44.48	45.08	44.18
Zn	0.61	0.84	0.43	0.61
$(FeAl)_2O_3$	0.10	0.05	0.07	0.07
CaO	0.21	0.01	0.08	0.02
SiO_2	0.17	0.11	0.12	0.14
CO_2	0.23	0.19	0.26	0.08
SO_2	0.96	0.12	0.68	0.44
On combustion yielded:				
H_2O	2.12	1.57	2.33	1.71
CO_2	3.62	2.96	3.22	2.92

The fume let down from the hoppers is spread over the floor in piles and set on fire with oil-waste. It burns for about ten hours and does not flame, but liberates a good deal of heat and some sulphur dioxide. The fine, loose blue powder is hereby converted into a porous, pinkish-white crust that is friable but sufficiently coherent to stand handling and charging. The roasted blue powder is free from carbonaceous matter and lead sulphide. An analysis showed the following percentages of composition: $PbSO_4$, 48.76; PbO, 46.82; Fe_2O_3, 0.32; Al_2O_3, 0.05; ZnO, 0.27; CaO, 0.48; SiO_2, 0.10; CO_2, 0.90; SO_2, 1.65; H_2O, 0.37—total, 99.72.

In refining the roasted blue powder in the slag-eye furnace, the object is to oxidize all the components of the charge as much as possible; hence little metallic lead is produced. To prevent any carbonaceous compounds from injuring the color, Connellsville coke is used as fuel.

The average daily charge for a furnace is made up of 2,800 lb. slag, 1,000 lb. blue powder, about 600 lb. dry-bone (carbonate ore), 450 lb. fume from the cooling pipes, and the necessary coke. For each pound of metallic lead about 1.6 lb. of paint are obtained, and the product of a furnace is 4,250 lb. of paint. Some crude galena has been added to the charge with satisfactory results.

In running the furnace it is important to have a hot top, a liquid slag, and no stoppages or irregularities, otherwise the paint obtained will be of inferior grade. Thus, when a furnace is started, the product of the first four or five hours is classed as

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blue powder, and that of the next 20 hours is still below the standard

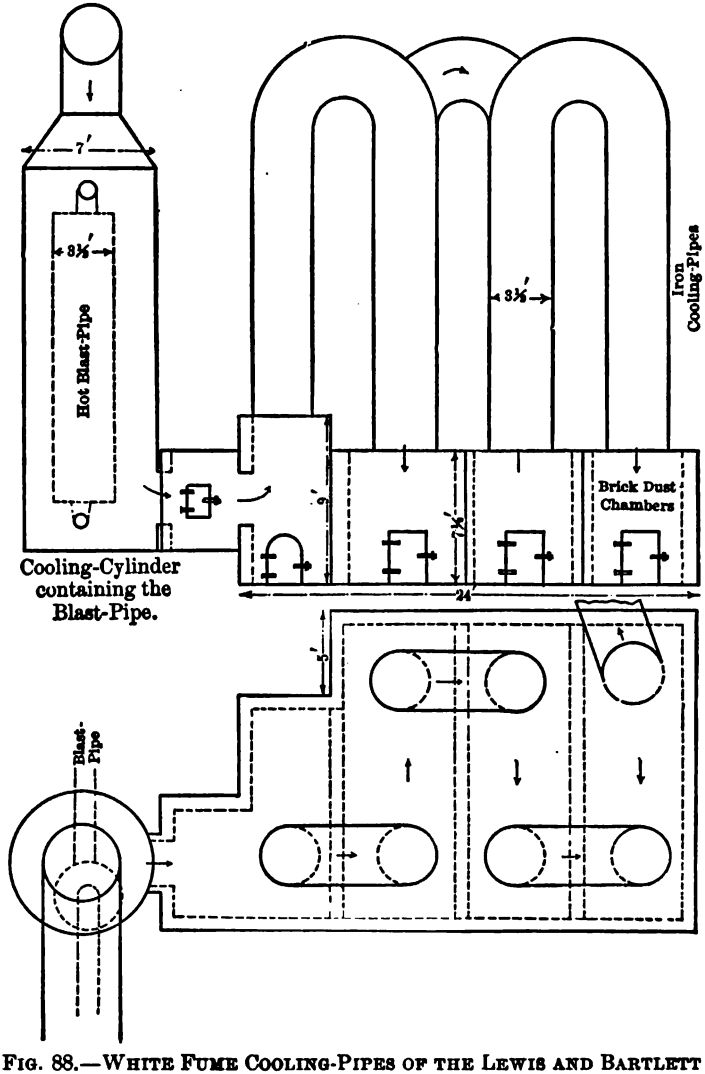


FIG. 88.—WHITE FUME COOLING-PIPES OF THE LEWIS AND BARTLETT BAG-PROCESS.

Plan and Elevation. Scale 1 inch = 10 feet.

A No. 5 Baker blower furnishes the blast for three slag-eye furnaces. From these the gases are drawn by a fan (6 ft. in diam-

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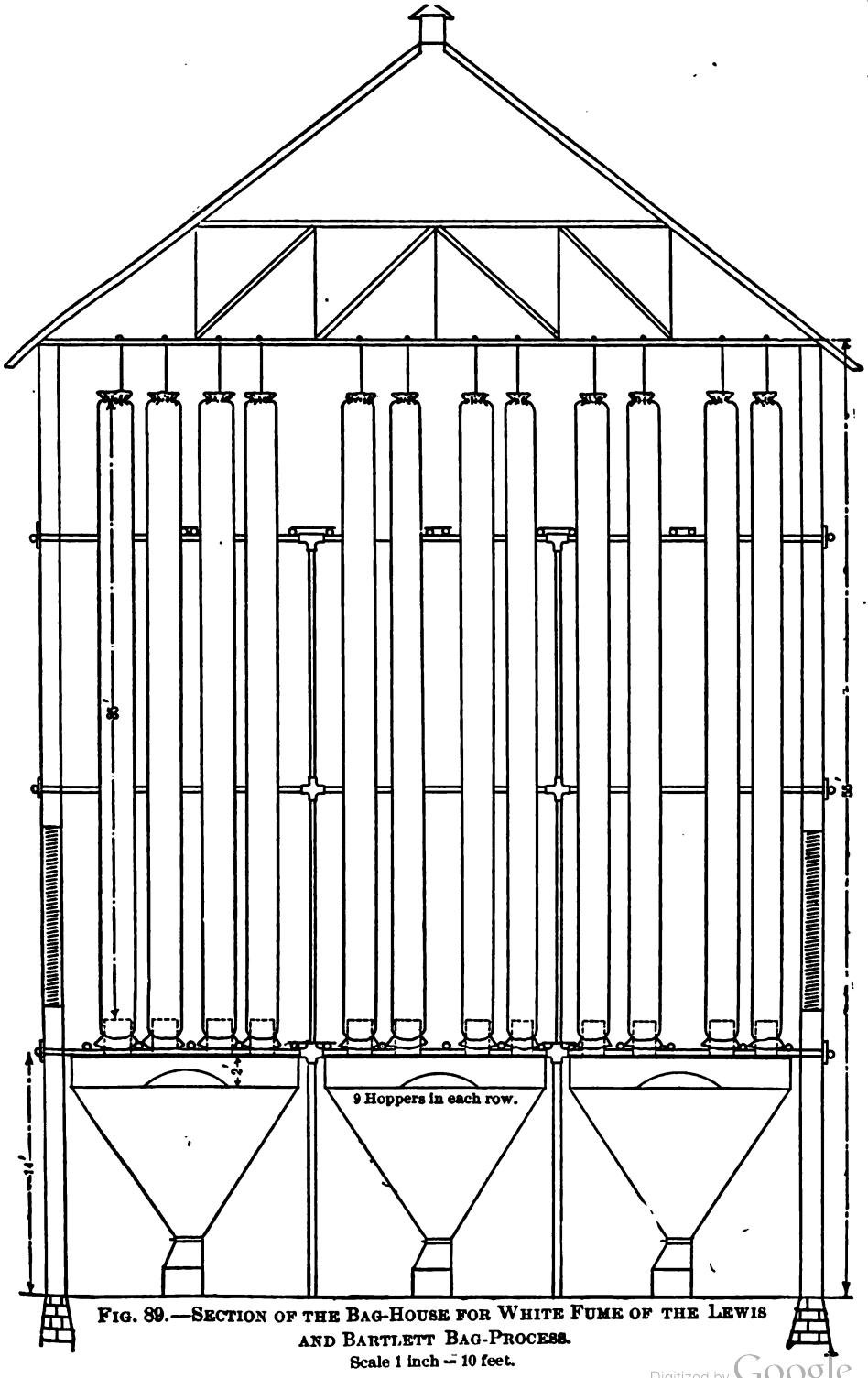


FIG. 89.—SECTION OF THE BAG-HOUSE FOR WHITE FUME OF THE LEWIS AND BARTLETT BAG-PROCESS.

Scale 1 inch = 10 feet.

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eter, 3 ft. wide, making 290 revolutions per minute), first through a set of cooling pipes which surround the blast-pipe, and then through a second set built over brick dust-chambers, where any heavy particles are collected to go back to the furnace, while the gases sufficiently cooled pass off into the second bag-room. The first set—shown in vertical section in Fig. 87 and connected with the second set in Fig. 86 (plan) and Fig. 88 (plan and section)—consists of two iron cylinders, 7 ft. in diameter and 20 ft. high, lined with firebrick and connected by a pipe 3½ ft. in diameter. The blast-pipe, 18 in. in diameter at its entrance near the bottom, widens to 3 ft., to take up more heat from the surrounding gases, and before passing out is again contracted to its normal size. The second set, four U-shaped iron cooling pipes, are 3 ft. in diameter and 20 ft. high.

The second bag-room or paint-house, shown in Fig. 89, is a brick building without any partition wall. It is 40 ft. wide, 90 ft. long, and 45 ft. high. It has two stories: the lower is 9 ft. high. The divisions and general arrangements are similar to those in the first bag-house. The lower floor, however, is boarded. The hoppers are made of wood and lined with iron. They are suspended by iron straps (12 in. apart) from 1½-in. pipes. These are laid across the 2-in. pipes at interval of 2 ft. Beneath the hoppers are wooden bins closed with canvas.

The paint from the bins is packed into barrels, each holding 500 lb. It has a good body, a good color, and mixes well with oil. It is also used in the manufacture of oilcloth, stained paper, and rubber goods. Its price in St. Louis in 1896 was $3.25 per hundred pounds, with pig-lead ranging from $2.43 to $3.03. Its composition is shown by the following analyses:

Insoluble	$PbSO_4$.	PbO.	ZnO.	Fe_2O_3.	CaO.	CO_2.	SO_2.	H_2O.	Total.
Per Cent	Per Cent	Per Cent	Per Cent	Per Cent	Per Cent	Per Cent	Per Cent	Per Cent	Per Cent
0.08	65.46	25.85	5.95	0.03	0.02	1.53	0.04	0.69	99.65
0.08	65.00	25.89	6.02	0.03	0.02	2.00	None.	0.85	99.89

§ 53. THE F. L. BARTLETT PROCESS FOR THE TREATMENT OF COMPLEX ZINC-LEAD SULPHIDES.* — The process carried out at

* While the process does not strictly belong to a treatise on lead metallurgy, extracting the precious metals rather by matting than by lead smelting, it has enough points in common with the hearth treatment of lead ores, with the collection of silver-bearing fumes by filtering, which is of daily growing importance in blast furnace work, and with other points of lead smelting, to more than justify its insertion in this book.

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Cañon City, Colo., and until lately at Portland,* Me., aims to oxidize and volatilize the zinc and lead of an ore, the resulting fumes to be saved and then refined to a pigment; to collect the precious metals in a copper-bearing iron matte and to dispose of the gangue in the form of a waste slag.

The Ore.—The ores commonly treated are an intimate mixture of silver and gold-bearing sulphides of zinc, lead, and iron, in varying proportions, with about 20% gangue. They are divided into two classes: those containing under 20% zinc to be smelted raw in the blast furnace and those containing over 20% zinc to be worked in the sintering or blowing-up furnace.

Sintering of Ore and Volatilization of Lead and Zinc.—The raw ore, crushed fine and mixed with bituminous coal, is charged on a perforated grate forming the bottom of an arched chamber red-hot from a previous charge; then under and over-grate blast are let on, which start the oxidation and raise the temperature and so change the loose sulphide ore into a more or less oxidized and sintered mass, while the fumes pass off through a flue to be cooled and collected.

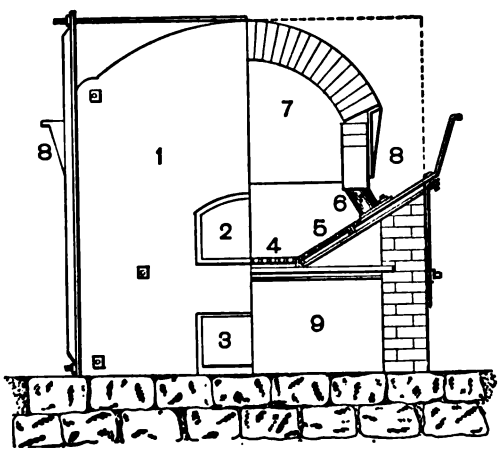
The furnace, called a "blowing-up furnace," is shown in Figs. 90 and 91. It consists of a perforated grate, 6 ft. long and 3 ft. 6 in. wide (4 and 5), resting on the side walls of the furnace and supported by cross-bars. It divides the arched chamber into the ash-pit, closed by the ash-pit door (3), and the hearth with its working-door (2) and flue (7). The roof is supported by channel irons (8), and its brick side walls (lately replaced by water jackets) rest on hollow columns (6), which are perforated on the hearth side to admit air under pressure on the charge. The ore mixture fills the charge-pockets (8) and passes between the columns (6) on to the hearth. The blast, entering at (10), is admitted under the charge through (4), at the side of it through (5), and on top of it through (6), thus insuring the desired degree of oxidation of the charge and especially of the fumes.

The ore containing over 20% of zinc is crushed through a 2-mesh sieve, mixed with from 15 to 20% slack coal and charged into the pockets. Supposing a previous charge to have just been

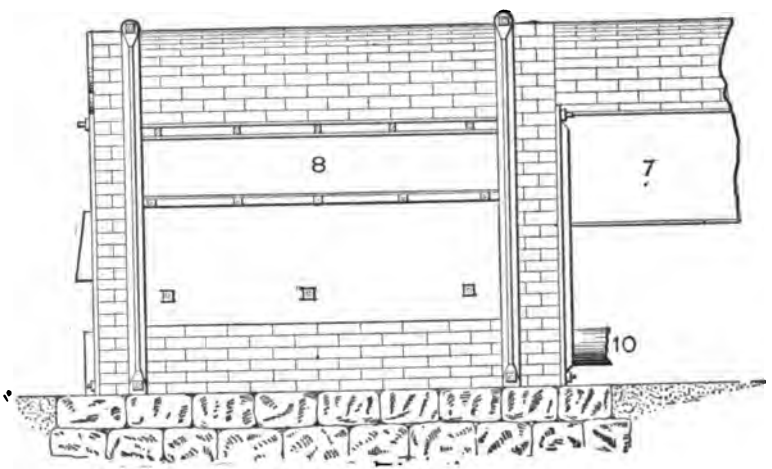
Bartlett, *Colorado State School of Mines Scientific Quarterly*, vol. ii., No. 1, p. 1; "The Mineral Industry," vol. v., p. 619; *Engineering and Mining Journal*, Aug. 8, 1899; July 1, Oct. 7, 1896; May 23, June 20, Aug. 22, 1896; *Ibid.*, Hofman, Oct. 28; Hawker, Dec. 9, 1893; *Ibid.*, Longmaid and Collins, June 26, Sept. 12, 1896; Private notes, 1896.

* Works burned down in 1895.

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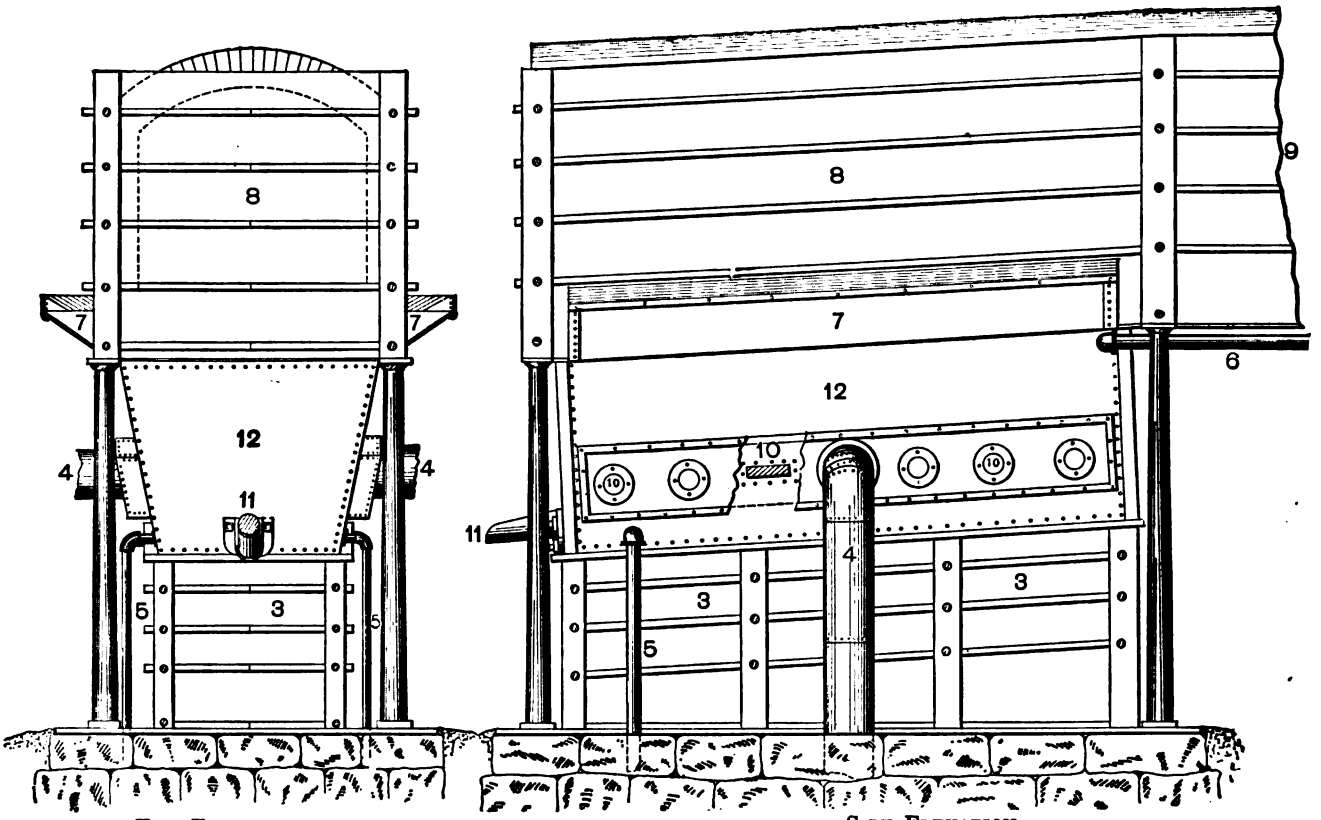
FRONT AND HALF SECTION.



SIDE ELEVATION.

FIGS. 90 AND 91.—BARTLETT'S BLOWING-UP FURNACE FOR ZINCY ORES.

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FIGS. 92 AND 93.—BARTLETT'S BLAST FURNACE FOR ZINC ORES.
Scale: 8″—1inch.

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drawn, the furnace to be red-hot, and the charge-pockets filled, the furnace-man will push the charge down on the grate, spread it to a depth of about 6 in. through the working door (2), start the blast and fill the pockets again. On arriving on the grate the charge will consist of ore and coke, as the coal has given up its volatile hydrocarbons while it occupied the lower part of the pocket and the upper part of the inclined grate (5). The coke ignites, and starts the roast-smelting of the charge; the flames, at first blue from carbon monoxide, soon become white from the lead—later from the zinc fumes; the temperature rises, the charge begins to clinker, and when the fumes have ceased to be evolved, about 30 minutes after charging, the blast is shut off and the clinker drawn out through the working door, when the furnace is ready for the next charge. Nearly all of the lead and most of the zinc have been driven off, and the precious metals are contained in the matte, which with the slag forms the clinker. Any parts of the charge that are not agglomerated are returned to the furnace and worked with the next charge. It takes from 20 to 40 minutes to treat a charge, the blast has a pressure of from 4 to 8 oz. per square inch, a furnace puts through about 6 tons in 24 hours, and one man attends to two furnaces. The composition of the clinker must, of course, vary somewhat; it contains less than 1% lead. The following four analyses show the other constituents:

	Per Cent.	Per Cent.	Per Cent.	Per Cent.
FeO	36.4	45.2	50.5	30.5
SiO_2 (a)	30.2	27.5	28.0	30.6
S	10.6	7.7	8.6	5.6
C	3.2	3.5	3.2	3.0
Zn	19.5	15.5	10.2	14.7
Cu	1.0	Trace.	Trace.	2.0

(a) Insoluble.

Smelting of Sintered Ore and Volatilization of Zinc.—The sintered ore is smelted in a low water-jacketed blast furnace with external crucible. The furnace is worked with a hot top in order to volatilize as much of the zinc as possible to be cooled and collected; its action is oxidizing rather than reducing; copper ore is added to assist in collecting the precious metals in the matte and to make a good separation of it from the slag. The furnace, 108 by 36 in. at the tuyeres, is shown in Figs. 92 and 93. In the figures, (3) represents the foundation walls well bound with iron,

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(4) the blast-pipe ending in an air-chest attached to the water jackets, the blast being superheated by being conducted through the flues, (5) the water-inlet, and (6) the water-outlet pipes, (7) the charge pockets, the charge sliding down the jackets, making an angle of about 45°, and being in the center only about 12 in. deep above the level of the tuyeres, (8) the shaft held together by corner irons and tie-rods, (9) the flue collecting the fumes from two furnaces and leading them to the cooling chambers and bag-rooms, (10) the tuyere openings, longitudinal slots, 8 by 1¼ in., the admission of air being regulated by a block pushed in or drawn out by a small iron rod attached to it, (11) the spout over which the melted masses flow into an overflow slag-pot, the bottom of the furnace slanting 1 in. to the foot from back to front, (12) the four steel water jackets, 40 in. high.

The furnace charge is made up by the addition of sulphide ores of zinc, lead and copper, and of some silicious ores to contain 17 to 20% zinc, 2½ to 4% copper, 3 to 10% lead, 15 to 20% sulphur, and 10% lime, the rest being silica and iron. The amount of coke used ranges from 6 to 15%, varying with the percentage of zinc that is to be driven off as fume. The furnace is run in such a way as to leave enough sulphur in the charge to bind all the copper and the remaining zinc and thus minimize the loss of silver by volatilization. The blast pressure is 12 oz. per square inch. The furnace puts through from 40 to 75 tons of charge in 24 hours, the amount depending upon the percentage of zinc present. Thus, with 20% zinc 40 tons are smelted, with 15% 50 tons, with 12% 75 tons. Three men attend the furnace on a shift. A furnace is blown out once a month, as it is necessary that often to clean out the flues that have become choked with dust. The matte produced, one ton from 10 to 12 tons of ore, runs about 40% copper and averages 125 oz. silver and 2 oz. gold per ton. The slags aimed at are silicious, as shown by the following analyses:

	Straight Iron Slags.			Iron and Manganese Slags.						Iron and Lime Slags.						
	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%
SiO_2 (a)	32.4	36.5	33.6	36.2	38.0	38.5	39.0	40.5	44.2	38.0	39.7	36.5	32.5	37.0	38.2	37.5
FeO	50.5	49.3	47.2	34.7	34.6	33.6	32.5	31.9	36.3	30.1	39.5	30.3	31.7	30.1	34.3	34.2
ZnO	15.2	12.5	14.2	13.0	14.2	12.5	11.8	11.2	7.2	11.4	10.1	9.3	9.3	8.2	3.7	3.1
CaG	....	....	2.2	4.0	2.5	4.6	4.2	4.0	2.0	15.4	7.8	18.5	18.5	19.4	20.4	19.2
MnO	....	....	....	7.8	8.1	7.0	8.0	8.9	7.4	3.1	....	2.0	4.2	1.3	3.0	2.1
Silver, oz.	2.5	1.75	2.0	1.5	1.0	.75	0.3	0.25	0.25	0.25	0.25	0.5	0.75	0.25	0.25	0.25

(a) Insoluble.

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Bartlett finds that zinc oxide is not present in the slag as a silicate, an opinion in which he does not stand alone. Basic iron slags dissolve as much as 20 and 25% zinc oxide. While the dissolving power decreases with the acidity of the slag, a certain proportion of lime aids in the solution of zinc oxide, but if it goes above 15% it hinders it. The slags are made to average from 7 to 8% zinc oxide, as a higher percentage drags silver into the slag. The silver contents ordinarily ranges from 0.25 to 1.25 oz. per ton, depending on the presence of copper and lime, the lowest figure referring to a charge with 4% copper, the highest to one with 1% copper, a slag high in lime being low in silver.

Cooling of Gases, Settling of Dust and Filtering of Fume.—The gases passing off in sintering the raw ore and in smelting the sintered product carry with them in the form of dust fine particles of the charge more or less oxidized, fumes of lead and zinc in the form of lead sulphite, sulphate and oxide and of zinc sulphide, sulphite and oxide, further sulphur dioxide, carbon dioxide, free oxygen and nitrogen. If they are allowed to expand and to cool, and are made to change their direction repeatedly and at the same time to strike upon large surfaces, they will readily drop the dust, while the fume can be satisfactorily collected only by filtering. It has been found that the silver carried off by the gases is contained mainly in the dust, so that with the dust well settled out the loss in silver by fume will be very small. The result is that under normal conditions the loss in silver is a fixed quantity, be the ore charge high or low in silver; thus the fume from the sintering furnace assays 5 oz. and from the blast furnace 8 oz. silver per ton. These figures will increase with oxidized zinc and lead ores on account of the lack of sulphur and also with arsenical and antimonial ores; they will decrease with sulphide zinc, iron, and copper ores, and increase slightly with sulphide lead ores and in the presence of lime; while, as stated above, the lime in the slag decreases the silver carried off by the latter. The total loss in silver in treating a charge containing 4% copper is $\frac{3}{4}$ oz. per ton, which may increase up to 3 oz. per ton if only 1% copper be present. There is a small gain in gold; the amount of lead collected in the fume shows a plus over that accounted for by the dry assay; the loss in zinc amounts to 5% and is accounted for by the assay of the slag.

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The gases from sintering and smelting are drawn off by Sturtevant fans placed back of a brick chamber having a vertical partition extending upward from the floor. The object of the chamber is to mix the gases from the sintering and smelting furnaces, and to equalize their temperatures. This is necessary, as the fans sucking off hot fumes would create a back pressure on those taking in cold fumes. In order to cool the gas mixture the fans suck in cold air. The cooled and diluted gases of approximately even temperature and composition are forced into an iron chamber. They enter at the bottom, are deflected by the roof and drop a large amount of dust. They then pass through a pair of oblong sheet-iron cooling flues, 8 by 3 ft. and 1,400 ft. long, which are supported in the air on a low roof-shaped trestle so that the dust, about 3% of the weight of the ore, that settles out may be easily removed through doors at the sides and raked into wheelbarrows or trucks. Generally 25 sq. ft. of sheet iron cooling surface are required for 1 sq. ft. of grate area of sintering furnace. At the ends of the cooling flues the gases, freed from dust, pass into one of the two bag-houses used alternately, where the fumes are separated out by filtering.

A bag-house, similar in construction to the one shown in Fig. 89, contains 1,500 bags made of loosely woven cotton or wool. The bags are 20 in. in diameter and 21 ft. long. Cotton bags, costing 7c. per yard, last from 18 to 24 months, and woolen bags, costing from 40 to 50c. per yard, last from 6 to 10 years. With cotton the temperature of the bag-room should not exceed 90° C., with wool not 120° C. Generally 200 sq. ft. of cloth are required per square foot of grate area of sinter furnace, and if twice cleaned or shaken in 24 hours, 1 sq. yard of woolen cloth will collect 1 lb. of fume per day, 1 sq. yard of cotton cloth ⅛ lb.

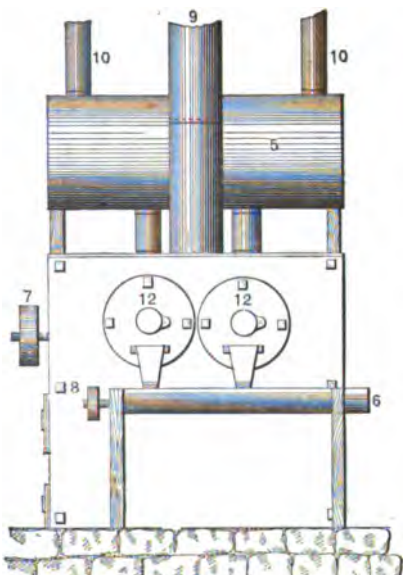
The raw fume averages:

$$12\ PbS,\ 30\ PbSO_4\ (+\ PbSO_3{*}) = 42\%.$$
$$14\ ZnO + 40\ ZnSO_3 = 54\%.$$
$$1\ C,\ 2\ SO_2,\ 1\ SiO_2,\ \text{etc.} = 4\%.$$

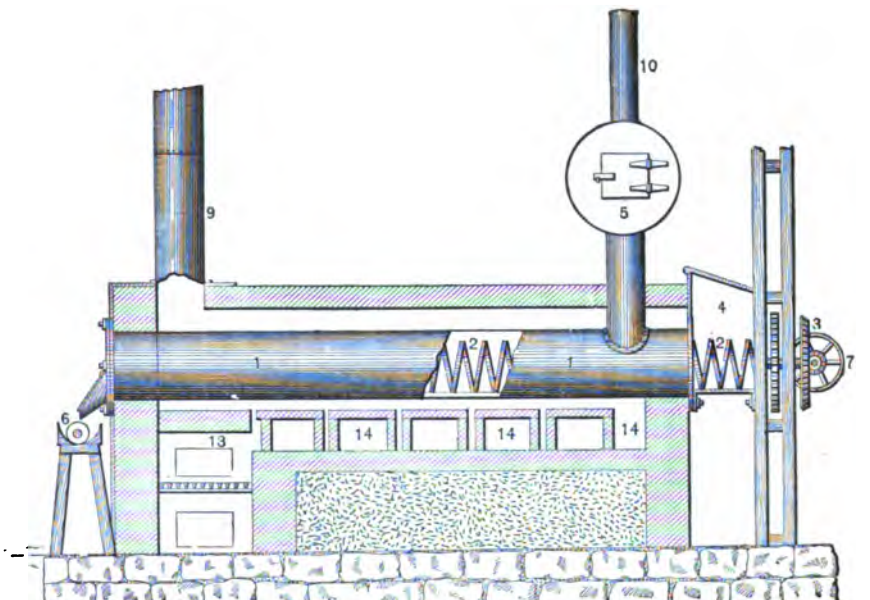
Refining Raw Fume into Refined Zinc-Lead Pigment.—The object of the refining process, which is an oxidizing muffle-roast, is to convert the lead and zinc compounds into lead sulphate and zinc

* Upon heating, $PbSO_3$ is decomposed into $PbSO_4$, PbS, PbO, and SO_2.

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END ELEVATION.



SIDE ELEVATION.

FIGS. 94 AND 95.—BARTLETT'S REFINING FURNACE FOR MAKING ZINC-LEAD PIGMENT.

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oxide, to eliminate finely divided carbon, arsenic, cadmium, and other impurities, and to compact the fume by grinding that it may be well adapted for use as a pigment. The furnace used for the purpose is shown in Figs. 94 and 95.

In both figures (1) represents a pair of cast-iron cylinders, 10 ft. long and 12 in. in diameter, (2) a heavy spiral screw with four longitudinal flat iron bars (not shown), the screw conveying the fume from feed to discharge and compressing it, the bars raising it and showering it through the air current passing through the cylinder, (3) a driving gear, (4) the feed-hopper, (5) the drum for collecting fine dust, (6) the screw conveyer for refined pigment, (7) the driving pulley, (8) the pulley of screw conveyer, (9) the chimney for products of combustion, (10) the chimney to carry off gases from the drum, (12) a cast-iron disk with circular air-inlet and discharge-spout, (13) the fireplace, and (14) flues for the products of combustion. Four cylinders refine all the fume of the plant, a cylinder treating from 1,200 to 1,500 lb. of fume per day. The temperature of the cylinders is kept at about 815° C. The pigment, passing in about 20 minutes through a cylinder, is discharged at (6) and then passed through a fine sieve which screens out any coarse particles. Four volumes of raw fume give one volume of refined pigment. This consists mainly of zinc oxide and lead sulphate with an excess of oxygen, the elementary analysis giving Zn 47.33%, Pb 24.92%, S 2.96%, Fe_2O_3, etc., 0.45%, O 24.34%.

It is bluish-white (white lead being yellowish-white), dense, and sells at 1c. a pound less than white lead.

The amount of dust collected in the drum (5) is small (1 lb. per ton of ore), and it contains many rare elements.

The plant of the American Zinc-Lead Co., of Cañon City, Colo., contains 12 sintering furnaces (six of which are always running), three blast furnaces (one being held in reserve), and treats about 100 tons of ore per day. The power required per ton of ore is about 3.25 horse power, and the labor 0.75 man. The power is distributed as follows: Five exhaust fans (6 ft.), 175 horse power; three underwind fans, 40; two Baker blowers, 30; one crusher and roller plant, 50; one electric light plant, 20; one refining plant, 10—or a total of 325 horse power.

The cost of treatment is not made public, but a treatment

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charge for galena-blende concentrates recently published by Bartlett* gives an approximate idea. His offer was:

21.2 oz. silver at 68.5c. per oz., less 5%		$13.78
0.05 oz. gold at $19 per oz.		.90
18% lead at 25c. per unit		4.50
26.8% zinc, nothing		0.00
Gross value		$19.18
Less treatment	$5.80	
Less freight	5.20	
		11.00
Net value		$8.18

* *Engineering and Mining Journal*, June 20, 1896.

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CHAPTER VIII.

SMELTING IN THE BLAST-FURNACE.*

§ 54. Introductory Remarks.—The treatment of lead ores in the blast furnace is generally discussed under three heads:

Precipitation,
Roasting and Reduction, } for Sulphide Ores.

General Reduction, for Oxidized Ores.

These processes are quite distinct in European practice, where large mines furnish certain smelting works with uniform ores. For instance, in the Harz Mountains, in Prussia, concentrated galena ores have been for years, and still are, smelted raw in the blast furnace. In other districts of Germany, as well as in France, the sulphide ores are always roasted before they go to the blast furnace, and oxidized ores are of such rare occurrence that they are hardly ever treated separately.

In the United States this regularity of treatment can seldom be pursued, as the ores treated by the smelter show the greatest variety, as may be seen from § 36 (Purchasing of Silver-Lead Ores). For this reason the chemical classification suited to European practice will be set aside and smelting in the blast furnace treated as one process, in which lead and silver are extracted from the ore in the form of base bullion by means of carbon, iron, and lime.

All smelting ores can be worked in the blast furnace; any ore containing over 4% of silica must be so treated. If it is a sulphide, it is generally first roasted in the reverberatory furnace; if a carbonate or a mixture of carbonte and sulphide, the carbonate prevailing, it is smelted at once. If the preceding

*Hahn, "Mineral Resources of the United States," 1882, p. 325: also *Engineering and Mining Journal*, Aug. 25, Sept. 1 and 8, 1883; Henrich, *Engineering and Mining Journal*, Oct. 3, 1883; July 17, 1886; Guyard, in Emmons' "Geology and Mining Industry of Leadville, Col., monograph xii., United States Geological Survey, 1886, p. 613; Hixon, "Notes on Lead and Copper Smelting and Copper Converting," New York, 1897.

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analyses of argentiferous lead ores of the Rocky Mountains and the Pacific coast be referred to, it will be seen that they usually contain much over 4% silica. This explains the universal use of the blast furnace in the West, and since the easy-smelting carbonate ores have grown scarce, being replaced by sulphide ores, more or less pure, the roasting department has become one of daily increasing importance.

§ 55. Roasting of Sulphide Ores in Pulverized Form.*—(*a*) *General.* In heating a metallic sulphide, MS., with access of air, metallic oxide and sulphur dioxide are formed:

$$MS + O_2 = MO + SO_2.$$

Part of the sulphur dioxide passes off, and part is converted in the presence of oxygen into sulphur trioxide by contact with the metallic oxide formed or with any indifferent substance; it derives its oxygen either from the air or from the metallic oxide which is reduced to a lower state of oxidation:

$$SO_2 + O + SiO_2 = SO_3 + SiO_2.$$
$$SO_2 + 2MO = SO_3 + M_2O.$$

The sulphur trioxide formed will either pass off or it will combine with metallic oxide to sulphate; it may have an oxidizing effect on metals and metallic compounds or be reduced to sulphur dioxide or even to sulphur by other reducing agents.

The combination of sulphur trioxide with metallic oxide to sulphate may in a general way be expressed by

$$MO + SO_2 = MSO_4.$$

Metallic sulphates upon heating are decomposed into metallic oxides and sulphur trioxide. Of the principal metallic sulphates, silver, iron, copper, zinc, nickel, cobalt, manganese and lead sulphate, silver sulphate,† is decomposed at a very low temperature, the other sulphates following with increase of temperature in the order given until lead sulphate is reached, which gives up sulphur trioxide only to a very small extent at a white heat. If it is to be completely decomposed the sulphur trioxide must be expelled from its combination by the stronger acid, silica (see §§ 6 and 8).

* Plattner, "Die Metallurgischen Röstprocesse," Freiberg, 1856; Balling, "Metallurgische Chemie," Bonn, 1868.

† Kerl, "Grundriss der Metallhüttenkunde," Leipsic, 1881, p. 70.

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The sulphur trioxide will either escape as such, especially in the presence of an excess of air, or it will be split into sulphur dioxide and oxygen if the decomposition took place in a confined space where the excess of air was wanting.

The oxidizing effect on a metal of sulphur trioxide, resulting from direct oxidation of sulphur, or from the decomposition of sulphate, is shown by

$$Ag_2 + SO_3 = Ag_2O + SO_2;$$

on a metallic sulphide by

$$FeS + 3SO_3 = FeO + 4SO_2,$$
$$Cu_2S + 3SO_3 = Cu_2O + 4SO_2,$$
$$Cu_2S + 4SO_3 = 2\,CuO + 5SO_2;$$

on a metallic oxide capable of being brought to a higher state of oxidation by

$$Cu_2O + SO_3 = 2CuO + SO_2.$$

Sulphur trioxide is further converted into sulphur dioxide by

heat alone : $SO_3 = SO_2 + O$ (see above),
sulphur vapor : $2SO_3 + S = 3SO_2$,
carbon : $2SO_3 + C = 2SO_2 + CO_2$,
carbon monoxide : $SO_3 + CO = SO_2 + CO_2$.

The reduction of sulphur dioxide to sulphur may be expressed by

$SO_2 + C = S + CO_2$. $\quad$ $SO_2 + 4H = S + 2H_2O$.
$SO_2 + 2C = S + 2CO$. $\quad$ $3\,SO_2 + 2CH_4 = 3S + 2CO + 4H_2O$.
$SO_2 + 2CO = S + 2CO_2$. $\quad$ $2SO_2 + C_2H_4 = 2S + 2CO + 2H_2O$.

(*b*) *Lead Sulphide,* PbS.—The behavior of lead sulphide was discussed in §§ 8 and 9.

(*c*) *Iron Sulphide,* FeS.—If iron sulphide is heated slowly with access of air to a temperature sufficiently high to start the oxidation it will at first be converted into ferrous oxide and sulphur dioxide:

$$FeS + O_2 = FeO + SO_2.$$

Part of the ferrous oxide will then be oxidized to magnetic oxide:

$$3FeO + O = Fe_3O_4,$$

while some of the sulphur dioxide will escape and some will be oxidized to sulphur trioxide:

$$SO_2 + O = SO_3.$$

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The sulphur trioxode which does not escape as such will peroxidize the magnetic oxide to ferric oxide:

$$2Fe_3O_4 + SO_3 = 3Fe_2O_3 + SO_2,$$

or combine with ferrous oxide to ferrous sulphate:

$$FeO + SO_3 = FeSO_4.$$

Ferrous sulphate upon increase of temperature will in passing through the stage of basic sulphate,

$$2FeSO_4 = Fe_2SO_6 + SO_2,$$

be converted into ferric oxide,

$$Fe_2SO_6 = Fe_2O_3 + SO_2,$$

and the sulphur trioxide set free will oxidize the undecomposed sulphide,

$$FeS + 3SO_3 = FeO + 4SO_2,$$

the products being the same as those of the first equation.

(*d*) *Iron Disulphide*, FeS_2*.—If heated without access of air iron disulphide gives up half of its sulphur, being converted into the monosulphide,

$$FeS_2 = FeS + S,$$

or rather the magnetic sulphide, Fe_8S_9. If roasted with access of air it is readily oxidized to ferric oxide, but since in roasting a charge the air cannot have free access to all parts of the pulverized mineral some of the sulphur passes off as vapor which burns to sulphur dioxide. The disulphide roasts more quickly than the monosulphide although it contains more sulphur. The reasons for this are that when the molecule of sulphur is distilled off it leaves the residual monosulphide porous, which offers more points of attack for oxidation, and there is a larger amount of sulphur trioxide set free with its strongly oxidizing influence. Iron disulphide occurs as pyrite and marcasite. The latter being, according to Brown,† more readily decomposed by atmospheric action than pyrite should also be more readily oxidized in roasting. But pyrite from different localities show a different behavior; thus pyrite from Sweden is perhaps the most difficult to roast dead, while that from southwestern Spain and Portugal represents the most free-burning.

(*e*) *Cuprous Sulphide*, Cu_2S.—Cuprous sulphide being readily

* Valentine, "Transactions of American Institute of Mining Engineers," xviii., p. 78.
† *Proceedings American Philosophical Society*, 1894, xxxiii., p. 225.

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fusible has to be heated very carefully if the roast is to be successful. It is at first converted into cuprous oxide,

$$Cu_2S + 3O = Cu_2O + SO_2,$$

and this by contact action into cupric oxide,

$$Cu_2O + SO_2 + 2O = 2CuO + SO_3.$$

Some of the sulphur trioxide set free escapes as such, some combines with cupric oxide to form cupric sulphate,

$$CuO + SO_3 = CuSO_4,$$

(which is decomposed again at an elevated temperature), and some is reduced to sulphur dioxide by acting upon undecomposed sulphide and upon cuprous oxide,

$$Cu_2S + 3SO_3 = Cu_2O + 4SO_2.$$
$$Cu_2O + SO_3 = 2CuO + SO_2.$$

The partial reduction goes on as long as considerable quantities of cuprous sulphide and oxide are present. This is the reason why roasted copper sulphide contains as much as 20 and 25% cuprous oxide after the sulphur has been expelled. Its conversion into cupric oxide has to be effected by the air.

(*f*) *Zinc Sulphide*, ZnS.—If heated with access of air, zinc sulphide is converted into oxide and sulphate:

$$2ZnS + 7O = ZnO + ZnSO_4 + SO_2.$$

In order to start the oxidation a higher temperature is required than with either lead, iron or copper sulphide. Zinc sulphide being infusible it can be heated to the elevated temperature required without any harm. The relative proportions of zinc oxide and zinc sulphate formed depend upon the temperature and the free access of air. The higher the temperature and the freer the access of air the more zinc oxide will result. The normal sulphate formed will be in part decomposed into the basic salt and sulphur trioxide, dioxide and oxygen being given off. The complete desulphurization of zincblende requires time, abundance of air, and a red-heat at the start, changing to a deep orange (1100° C.) toward the end, to decompose the basic sulphate. All blendes do not show the same behavior in roasting. Thus in the European zinc centers it has been found* that blendes from the older formations of Scandinavia give up their sulphur less readily than do those from the later formations of Styria (Austria). Also,

* Jensch, *Berg- und Hüttenmännische Zeitung*, 1894, p. 299.

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the dark blendes, rich in iron, are more difficult to roast dead* than the light-colored varieties. Jensch found that the sulphur in roasted iron-bearing blende is present as iron sulphide, which is surprising, as this is so much more easily oxidized than zinc sulphide.

(*g*) *Manganese Sulphide*, MnS.—Heating manganese sulphide at a moderate heat with access of air converts most of it into manganous sulphate and the rest into mangano-manganic oxide; the sulphate at a clear-orange heat is converted with evolution of heat into mangano-manganic oxide; sulphur dioxide, sulphur trioxide and oxygen being given off.

(*h*) *Matte*, PbS, FeS, Cu_2S (ZnS, MnS).—The behavior of matte in an oxidizing roast will vary somewhat according to the relative amounts of lead, iron, and copper. Of the three leading sulphides, iron sulphide is first oxidized, then follow cuprous and lead sulphide successively. The more lead and copper a matte contains the lower is its melting point, hence the roasting of a matte produced in a lead blast furnace must be begun at a low temperature and progress slowly. As with an increase of temperature the normal ferrous sulphate is changed into the basic salt, this being decomposed at about the same temperature at which cupric sulphate is formed, the sulphur trioxide set free assists very much in oxidizing cuprous sulphide and oxide. The decomposition of cupric sulphate with increase of temperature will assist in oxidizing any zinc and manganese sulphide present. Owing to the tendency of matte to sinter, even when partially roasted, much zinc and manganese will be present in the roasted matte as sulphate.

(*i*) *Silver Sulphide*, Ag_2S.—If silver sulphide is given an oxidizing roast, it is converted into finely divided metallic silver and sulphur dioxide,

$$Ag_2S + 2O = 2Ag + SO_2,$$

which is attended with loss. If present in small quantities in other metallic sulphides, the sulphur trioxide set free in their decomposition has a tendency to sulphatize the silver:

$$2Ag + 2SO_3 = Ag_2SO_4 + SO_2,$$
$$Ag_2S + 4SO_3 = Ag_2SO_4 + 4SO_2.$$

This is especially the case with the basic sulphate of iron and the

* Minor, *Berg- und Hüttenmännische Zeitung*, 1889, p. 466.

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sulphate of copper decomposed at a temperature when most of the metallic sulphides have been oxidized. The decomposition of silver sulphate, whether caused by heat or by the presence of oxides, such as magnetic oxide of iron or cuprous oxide,

$$Ag_2SO_4 = Ag_2 + SO_3 + O,$$
$$Ag_2SO_4 + 4Fe_3O_4 = Ag_2 + 6Fe_2O_3 + SO_2,$$
$$Ag_2SO_4 + 2Cu_2O = Ag_2 + 4CuO + SO_2,$$

is attended with a great loss of silver. In roasted ore the silver will be found as undecomposed sulphide, as sulphate, and in the metallic state.

§ 56. The Operation in General.—In order to decide whether it is necessary to roast an ore or whether it can be smelted raw, the character and amount of sulphides, the richness of the ore, and the cost of roasting have to be considered. As a general rule, any ore containing 12% sulphur is best roasted before it is smelted. To some extent the richness of the ore in silver and its coarseness may modify this rule. On account of the loss in silver suffered in roasting, it may in some cases be better to smelt an ore raw which contains more than 12% sulphur. An ore running 100 oz. silver to the ton is rarely roasted; some metallurgists* draw the line at 50 oz. silver, which, however, seems rather low. Gold-mill concentrates with 10% sulphur are always roasted.

Pure argentiferous galena ores rarely come in such quantities to a smelter as to make their separate treatment necessary. They are generally added to the charge in the raw state in such quantities as to make up for the common deficiency of lead. Impure galena ores are usually mixed before roasting with sulphuretted ores that are free from lead. This is for two reasons: to furnish the required lead for the blast-furnace charge and to reduce the percentage of sulphur in the roasting charge. If the charge contains over 20% sulphur it is liable to become sticky at the beginning of the roast, and this must be avoided if the sulphur is to be satisfactorily eliminated.

In smelting pyritic ores in the blast furnace pyrite consumes iron (see § 67 *h*) and thus reduces the capacity of the furnace for ore; it forms much matte, consequently a considerable percentage of lead and silver is not directly recovered in the form of base bul-

* Newhouse, *Engineering and Mining Journal*, Feb. 28, 1891.

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lion. The matte has to be roasted before resmelting. The iron previously consumed in the blast furnace becomes again available, so that the actual consumption of iron in smelting ores raw is not so great as is generally assumed. Whether a pyritic ore shall be roasted or not is decided by the percentage of sulphur it contains above that which is required for the amount of copper present. A large quantity of matte, is, however, to be avoided, since the silver entering the slag increases with the percentage of matte formed. This bad effect begins to show itself with 10% of matte in the charge, and every smelter tries to keep the matte-fall under 5%.

Blende has a very deleterious influence in the blast furnace (see § 69 *j*). If it is present to any extent the ore will have to be roasted, for if it is smelted raw a large percentage of slag will necessarily have to be added to the charge to diminish the relative amount of blende and thus reduce its bad effect. In a general way it may be said that the higher the percentage of lead in a blast-furnace charge the more blende is permissible. For instance, if with mixed sulphide ores containing little or no pyrite the amount of lead present is twice that of zinc the ore is smelted raw; if zinc and lead are present in equal amounts, or if there is more zinc than lead, it is best to roast the ore before smelting. As the zinc sulphate formed in roasting blende requires a deep-orange heat and a considerable time to be converted into oxide, ores rich in blende are best roasted separately from those containing little of it.

The results obtained in roasting an ore depend not only on its chemical composition but also upon the size to which it has been crushed, the thickness of its bed in the furnace, the amount of rabbling it receives, the time it remains in the furnace, and the temperature to which it is exposed.

As galena oxidizes but slowly when heated with access of air, a large number of surfaces (fine-crushing) are necessary, if the roast is to have the desired result. Then, as the roasting proceeds in each particle from the surface to the center, it is probable that if the galena is too coarse a reaction may take place where the oxide and sulphate formed at the surface come in contact with undecomposed sulphide at the center, and the resulting metallic lead will cause a considerable loss in lead and silver—another reason for fine-crushing. Ores that do not roast readily,

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i.e., ores rich in galena and blende, are crushed through an 8-mesh sieve. Ores that roast easily, *e.g.*, pyritic ores and iron matte with 10% lead, are crushed through a 4-mesh sieve. The oxidation with these is rapid, and the roasted product less fusible and more porous than it would be if richer in lead.

The thickness of the charge on the hearth and the amount of necessary working depend also upon the character of the ore. The richer it is in lead and zinc the thinner must be the charge and the more work will be required. The thickness of the bed varies from 3 to 6 in., the rabbling being repeated from every ¾ to 1½ hours.

The time required to roast the ore depends upon the readiness with which it oxidizes. Ores in which galena prevails require a very slow roast and a low temperature throughout, as even with the most careful roasting it is impossible to prevent the roasted ore from retaining undecomposed lead sulphide. This is also the case with matte rich in lead and copper. Pyritic ores can be roasted quickly, and there is no danger of the half-roasted ore becoming sticky and adhering to the hearth of the furnace. Ores rich in blende require a considerable time and a high temperature if the zinc sulphide is to be completely converted into sulphate and this fully decomposed.

The temperature at which the roasted ore is to be withdrawn from the furnace must be regulated by the character of the sulphates formed and by the fusibility of the charge which can be taken out as a pulverulent, a sintered, or a fused mass. As regards the subsequent smelting, it is best to slag the ore, as by obtaining the roasted ore in lump form the disadvantages of treating fine ores in the blast furnace are overcome; but other considerations often prevent this. The principal ones are the loss in lead and silver and the increase of cost. Newhouse gives as a result of a series of experiments in roasting ores containing from 12 to 18% lead a loss of from 15 to 18% lead and of 2% silver with subsequent fusion, and from 2 to 5% lead and none in silver without it. By agglomerating the ore the loss will be only slightly higher than when it remains pulverulent. The sulphur will not be so effectually removed as when the ore is slagged, but more so than when it remains a powder. For instance, slag-roasted ore contains from 1 to 3% sulphur, while roasted pulverulent ore retains from 3 to 7% sulphur. The loss

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increases on the whole with percentage of lead in the charge. It used to be said that an ore with 10% lead or less could be safely slagged; with from 10 to 20% only agglomerated; if the lead ran over 20% the temperature was kept so low that the roasted product remained pulverulent, or only slightly adhesive when drawn from the furnace. This referred to the mixed sulphide ores treated by Western smelters, which as a rule run low in lead and high in silver. It is not applicable to the pure galena concentrates free from silver (as in the Mississippi Valley) or low in silver (as in most European silver-lead works) because in both instances the ores are always slagged to a greater or less degree. The charges running 50 and 60% lead and free from impurities require when roasted a very slight increase of temperature to be slagged, care being taken to keep it as low as possible. Therefore the loss in lead and silver is slight, although the percentage of lead is high.

To-day the fusion of roasted ore has been given up in many instances, as recent investigations have shown that the loss in silver is higher than was once supposed, and, so far, at least, no satisfactory methods exist for condensing the fumes issuing from a roasting furnace. The great improvements made in bricking finely divided ore by machinery (§ 94) have removed one of the disadvantages of having to use roasted ore in the blast furnace that has not been sintered or slagged; but the imperfect elimination of sulphur remains.

§ 57. Roasting Furnaces in General.—The roasting of lead ores can be carried on in heaps, stalls, kilns, and reverberatory furnaces. So-called mixed ores, consisting mainly of galena, pyrite, chalcopyrite, and blende, and containing comparatively little gangue, are sometimes roasted in heaps and stalls, the sulphur-dioxide being allowed to go to waste; or in kilns, when the sulphur dioxide is to be converted into sulphuric acid. As this roasting is comparatively rare with lead ores, but very common with copper ores, and as the apparatus is practically the same, the method is best omitted here and the reader referred to the works on the metallurgy of copper* and the manufacture of sulphuric acid.† Here only the roasting in the reverberatory furnace will be considered.

* Howe, "Copper Smelting," Bulletin No. 26, U. S. Geological Survey, Washington, 1885; Peters, "Modern Copper Smelting," New York, 1895.

† Lunge, "Sulphuric Acid and Alkali," vol. i., London, 1891.

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In order to roast an ore in the reverberatory furnace it is generally introduced at the cool flue-end and slowly moved down the hearth to the discharge, which is usually the hottest part of the furnace. During its stay in the furnace it is turned over more or less frequently in order to expose new surfaces to the oxidizing action of the heated air current passing over it. In describing the furnaces and their work it will be convenient to classify them as hand and mechanical reverberatories, according to the manner of moving and stirring the ore.

With the mechanical furnaces, the hearth can be either stationary or moving. The leading furnaces used in the treatment of lead ores will be discussed, as well as a few others so far confined to the roasting of pyritic ores free from or low in lead.

I. Hand Reverberatory Roasting Furnaces.
   The Long-bedded Furnace.
II. Mechanical Reverberatory Roasting Furnaces.

   (*a*) With Stationary Hearth:
      The Ropp Straight-Line Furnace.
      The Pearce Turret Furnace.
      The Brown Horseshoe Furnace.
      The O'Hara Furnace.
      The Keller Automatic Roaster.
      The Wethey Furnace.
   (*b*) With Moving Hearth:
      The Brückner Furnace.

Until within a few years hardly any mechanical furnaces were used for the roasting of lead-bearing ores. The enormous advances made, however, in the roasting of copper ores has had its effect upon lead smelters, and many mechanical furnaces are running to-day; but their work has not been as satisfactory as with pyritic ores and a somewhat justifiable reaction has set in against them.

In order to reduce the sulphur contents of a lead-bearing ore to a satisfactory degree, it must be possible to accelerate and retard the progress of the ore from feed to discharge as the conditions may require, and toward the end of the roast, when the temperature is brought to the highest permissible point, to rake and turn over the charge much more frequently than at the beginning. In a mechanical furnace with stationary hearth this rak-

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ing and turning over is uniform, as it depends on the speed of the rakes. If these travel too slowly the charge will not be sufficiently worked near the discharge; if too quickly the ore near the feed will be turned under, while the sulphur on the surface is still burning off, with the result that the ore will begin to cake and form lumps. As this must be avoided, the mechanical furnace has to run slowly, and the ore not being sufficiently stirred will retain a comparatively large amount of sulphur. The relatively small quantity of ore the mechanical furnace puts through when it has to reduce the percentage of sulphur to say 3% very often does not warrant the expense of putting it up and keeping it in repair. It has therefore been suggested (and has been done in a few instances) to rough-roast the ore in a mechanical furnace and to finish it in a small hand reverberatory furnace.

Another difficulty is that of the caking of the ore on the hearth bottom and obstructing the passage of the stirrers. This has been overcome to a considerable extent by roasting, not on the brick hearth as in the hand reverberatory furnace, but on a bed of 2 or 3 in. of dead-roasted pyrite or any substance (hematite, limestone, etc.) that does not readily combine with the roasting charge, and by plowing up or breaking up by other means any crust as soon as it forms. Finally, it has been found that the blades (plows and rabbles) of the stirring apparatus last a very much shorter time in roasting lead-bearing ores than pyritic ores. This must be caused by a chemical effect of the ore, as there is no reason why the mechanical wear should be greater with the lead-bearing ores. It has been suggested that lead sulphide may be to some extent decomposed by the red-hot blades, but this, as far as the writer is aware, has not been proved. Should it be the case, the loss of lead and silver by volatilization in a mechanical furnace having rakes ought to be very much greater than in one having a revolving hearth. With furnaces of this class the rabble arms are always in pairs when the rabbles have the form of plates set at an oblique angle to the arm. The rabbles of one arm are set in an opposite direction to those of the corresponding one, that the ore may be uniformly distributed over the hearth and prevented from accumulating on one side or the other.

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In mechanical furnaces having a revolving hearth, of which the Brückner furnace is the only one in use, the ore can be turned over quickly or slowly by varying the number of revolutions; further, it can be kept any length of time exposed to the oxidizing heat of the flame and gases. Ores containing rather a high percentage of lead are roasted in the Brückner furnace, but its tonnage is small; there is also much danger of the ore adhering to the wall and the percentage of fluedust formed still remains great.

§ 58. THE LONG-BEDDED HAND-ROASTING FURNACE. — This roasting furnace, the German *Fortschaufelungsofen*, has survived all modifications. It has a single roasting hearth, from 40 to 80 ft. long and from 14 to 17 ft. wide, and working doors on either side through which the charge, fed at the flue end, is slowly worked down to the bridge end and there discharged.

The old furnaces, about 6 ft. wide, with working doors only on one side, are probably not to be found now. Furnaces having two hearths, one on top of the other, can still occasionally be met with—for instance, at the large silver-lead works at Mechernich,* Prussia. The advantage of a double hearth is that the longitudinal extension is only one-half as great as with one hearth, and therefore, if cramped for space, one may be justified in erecting it, though a double hearth requires a much more solid construction, and is therefore more expensive; then if any repairing has to be done on the lower hearth, which is often the case, that part of the upper hearth situated above the place to be repaired has to be torn out to permit work. Finally, with a double hearth the workman, to turn over and move the ore on the upper hearth, has to stand on a high truck that runs on rails parallel with the furnace. Standing on a shaky platform the man cannot do as good work as when he is on solid ground, and it is difficult to inspect his work. The consequence will be that all of the ore is not moved toward the firebridge; particles will remain behind, and if the charge be rich in lead it will adhere to the hearth and gradually form a crust, which will have to be cut out. This necessitates shutting down the furnace in order to build a wood-fire on the hearth near the crust, so that the flame may pass over and soften it. It is claimed that fuel is saved by a double

* *Berg- und Hüttenmännische Zeitung*, 1875, p. 129; 1886, p. 434; *Engineering and Mining Journal*, March 8, 1877.

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hearth, as much less heat is lost through the roof than is the case with a single hearth, but the same advantage can be gained by placing a layer of sand on the roof of a single-hearth reverberatory.

There are three modifications or kinds of hearth: the level hearth, the slagging hearth (fuse-box), and the sinter hearth.

Figs. 96 to 101 represent a *Long-bedded Hand-roasting Furnace with a Level Hearth.* It is one of the furnaces contained in the roaster building shown in Fig. 154. The hearth is 40 by 14 ft. A furnace of this form is used if the charge is not to be sintered or fused, but withdrawn near the bridge in a pulverulent state. As the temperature is kept comparatively low at the bridge, 40 ft. may be sufficient length to utilize all the heat generated, although the furnace is commonly made longer.

Special attention is called to the vaulted arches which support the hearth, the absence of offsets (the hearth having a gentle rise from bridge to flue), the slope from the center to the doors, the discharge openings for roasted ore, and the damper in the flue. The distance between the working doors (51 in.) is less than is usually found; the width of the fireplace (21 in., Fig. 101) proving insufficient, it was increased to 36 in. (Fig. 96), and to supply some of the room required for this, the firebridge (27 in., Fig. 101) was made narrower (22¼ in., Fig. 96) and received air-flues.

The horizontal grate of the furnace is replaced in many instances, especially with dry coal, by a step-grate. The Bartlett blowing-up furnace (Figs. 90 and 91) is doing effective work in burning slack coal under boilers, and should do the same in connection with roasting furnaces, as it permits perfect combustion, and any excess of air, thoroughly superheated, can be made to pass off with the products of combustion. At some works, oil, when sufficiently low-priced, has become a valuable fuel for roasting furnaces, as the excess of air required to atomize the oil, over what is necessary for combustion, furnishes a strongly oxidizing atmosphere. Finally, producer gas, generated by the Taylor or Wellmann furnaces, is used in a few works, and promises to grow in importance.

The work done in the long-bedded hand roasting furnace is shown by three examples in the subjoined table:

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Fig. 100.



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LONG-BEDDED HAND-ROASTING FURNACE WITH LEVEL HEARTH.

	I.	II.	III.
Length of hearth	60′	66′	75′
Width of hearth	14′	16′	14′
Hearth area, square feet	840	1,056	1,150
Length of grate	8′	7′ 9″	8′
Width of grate	3′ 4″	2′ 6″	3′ 6″
Grate area, square feet	14.6	19.4	28
Ratio hearth to grate area	57½ : 1	54½ : 1	41 : 1
Space above firebridge, length and width	7′ 9″×2′ 2″	7′ 9″×2′ 2″	2′ 6″×1′
Space above fluebridge, length and width	No fluebridge.	4′ 2″ ×8″	No fluebridge.
Height of firebridge above hearth	14″	12″	20″
Height of roof above firebridge	18″	20″	12″
Height of fluebridge above hearth	.....	6″	.....
Height of roof above fluebridge	.....	15″	.....
Depth of grate below top of bridge	14″	15″	17″
Character of ore	¾ Pyrite. ¼ Galena.	Matte. Concentrates.	Pyritic galena.
Composition of ore	(*b*)	.....	(*a*)
Screen-size of ore (mesh)	2	12 and under.	2
Depth of charge near fluebridge	3–4″	.....	5″
Ore stirred every (minutes)	30	20	40
Roasted ore drawn every (hours)	8	4	4
Time ore remains in furnace (hours)	22	24	24
Tons of raw ore in 24 hours	8.1	12	9
Pounds of ore roasted per square foot of hearth area	20	21.8	15.65
Character of roasted ore	Partly sintered.	Pulverulent.	Partly sintered.
Per cent. sulphur in roasted ore	1.2 (*c*)	2–5	3

(*a*) SiO_2 16–5, Fe 28–25, Pb 12–25, Cu 2–3, Zn 4–6, S 33, Ag 25–40 oz., Au 0.08–0.30 oz. (*b*) Pyrite: Fe 37, SiO_2 5, Cu 1, Zn 4–5; Galena: Pb 45–60, Zn 10–12. (*c*) Best roast.

The amount of fluedust formed is very small, less than 1% of the ore charged. The fuel consumed is about two tons. The labor required varies with the tonnage, a fair figure being two men on a 12-hour shift, with a 60 by 14 ft. furnace, for 9 tons of raw ore. If the furnace is 80 ft. long, and puts through 12 tons, an additional man will be required.

If ore is to be slag-roasted in a furnace with level hearth, this has near the firebridge a slight depression or sump in which the ore is melted down. The consequence of this is that the ore in front of the third and fourth doors from the firebridge is pasty, and if the fire has not been carefully watched the heat may be excessive up to the fifth door. Not only does this interfere with a good roast, but it also renders the moving of the ore with the paddle a very arduous piece of work. To counteract this, it has been and often still is the custom with such furnaces to collect the ore from the third and fourth doors in a heap in front of the second door, and to melt it down into the sump, whence it is removed through the first door.

A furnace of this description is used at Mine La Motte,* Mo.,

* Private communication from J. T. Monell, May, 1891.

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where galena concentrates are slag-roasted. The hearth, 55 by 11½ ft. on the inside, is slightly inclined from the flue to the bridge, making the respective distances from the hearth to the horizontal roof 15 and 22 in. The top of the bridge-wall 22½ in. wide, is 9 in. above the hearth and 13 in. below the roof. The grate, 10 ft. by 21 in., is 3 ft. 6 in. below the top of the bridge, this depth being necessary on account of the fuel used, which is wood. Of special interest is the construction of that part of the hearth where the ore is fused and of the bridge. The former is built into a wrought-iron pan resting on brick pillars between which the air circulates freely. It is formed by a full course of firebrick, and is slightly concave. The air-cooling has proved very effective in preventing the corrosion of the brick. A similar result is produced by the air-flue passing through the bridge. When this was at first constructed in the usual way it was found that the middle part of the bridge was apt to be eaten through by the slagged ore. As a central wall divides the fireplace into two parts, the idea was conceived of closing the air-flue in the middle and erecting a small chimney on the roof, communicating with both parts of the air-flue. By this means a strong current of air could be passed through the flue. The experiment was a success, and the improvement has now been used for over 15 years.

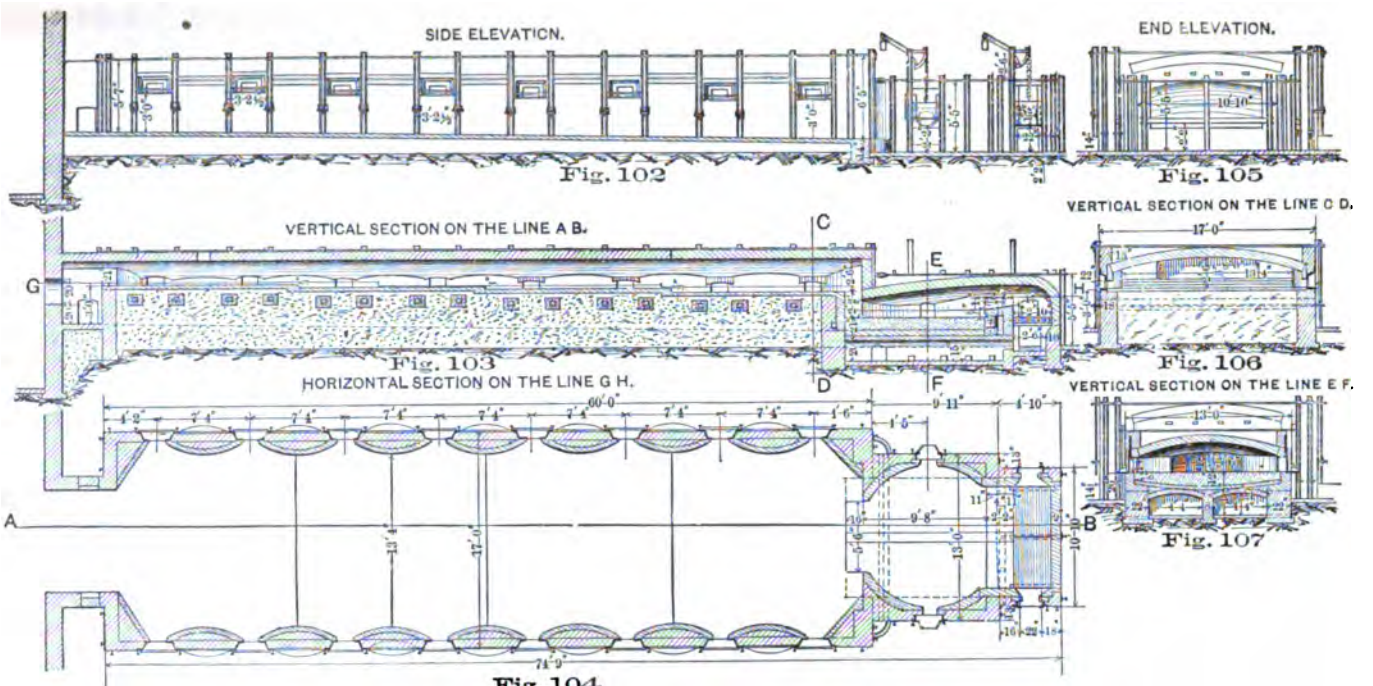
The writer has used a water jacket in fusing antimoniate of lead in a reverberatory furnace, and has found that it stopped all leakage at the bridge.

The Mine La Motte furnace is charged every 6 hours with 2 tons of galena concentrates, to which some sand is added as acid flux. The thickness of bed is 6 in. The galena runs from 40 to 70% lead, and from nothing to 25% iron, and is crushed to pass a 12-mesh sieve. The slagged ore retains from 4 to 6% sulphur; four men work on a 12-hour shift, and 0.42 cord of wood is burned per ton of ore.

The furnace at Bonne Terre,* Mo., is similar to the one at Mine La Motte. The roasting hearth is 40 ft. long by 11 ft. wide, the sinter hearth 11 ft. square and 8 in. deep. The grate is 7 ft. by 2 ft. 6 in., the firebridge 20 in. above the grate and 16 in. above the sinter hearth, and the roof 12 in. above the bridge. At the flue-end of the furnace is a bridge 8 in. high and the roof is 16 in. above the fluebridge. The ore treated is a galena con-

* Private communication from G. Setz, March, 1897.

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FIGS. 102 TO 107.—LONG-BEDDED HAND-ROASTING FURNACE WITH SLAGGING-HEARTH AND FUSE-BOX.

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centrate, 5 mm. and smaller, with Pb 72%, Fe 4%, (CaMg)O 5%, S 15%. It is charged to a depth of 4 in., stirred on the roasting hearth every half-hour, on the sinter hearth every hour, and drawn every six hours. It remains 30 hours in the furnace and retains 3.5% sulphur. The furnace roasts 5 tons of raw ore in 24 hours or 22.5 lb. per square foot of hearth area.

A Long-Bedded Hand-Roasting Furnace with Slagging Hearth (Fuse-Box) is represented in Figs. 102 to 107. The main improvement of the furnace, first erected at the Omaha and Grant Works, Denver, Colo., consists in carrying on the roasting on a hearth separate from that on which the slagging or fusing takes place, the ore being made to drop through a vertical flue, from 22 to 24 in. high, on to the slagging hearth. The hearth, as seen in Fig. 104, has the form of a reverberatory smelting furnace. In order to obtain the desirable sudden change from the high temperature in the slagging hearth to the low temperature on the roasting hearth, the flue-space above the latter is very much enlarged. Thus the flame on leaving the slagging hearth (Figs. 103 and 104) passes through a flue 5 ft. 6 in. by 1 ft. 4 in., which on entering the roasting hearth is suddenly enlarged to 17 ft. by 2 ft. 6 in. The sudden increase of area causes a correspondingly sudden decrease in temperature, and this produces the sudden change from pasty or fused to powdery ore. A detailed description of the furnace is not necessary, as the drawings can be understood without it. A few remarks, however, may be in place.

The roasting hearth is in four separate planes, divided by 3-in. offsets, which serve to keep the charges apart. The distance between roof and hearth is thus diminished by stages, leaving the former horizontal. This can also be done with a single inclined hearth, which is preferred by Hodges,* who gives it as his experience that the offsets furnishing points of attacks, lead to the injury of the hearth, as shown in Fig. 101, and are not required to separate one charge from another. Only that part of the roof above the lowest roasting hearth is built of firebrick, the rest is of red brick. In the end view (Fig. 105) are seen four openings for admitting air into the roasting hearth. The additional air required enters through the two doors next to the flue, which leads into the fuse-box, the door-lids being left slightly ajar.

* *Engineering and Mining Journal*, Oct. 24, 1885.

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The working bottom of the fuse-box used to be (and is still sometimes) made of quartz sand, to which small amounts of slag are added after the sand has been put into the furnace and heated until it becomes slightly sintered on the surface. This bottom is represented in the drawing. It has not proved as satisfactory as was expected, and has been generally replaced by a 9-in. firebrick bottom, built slightly concave. The bottom (Fig. 107) rests on two arched roofs, and is thus cooled by air circulating below it. In the firebridge (Fig. 103) there is on one side of the air-space a heavy cast-iron bridge-plate to bear the longitudinal stress of the hearth. The parts of the furnace that wear out fastest are the flue leading from the fuse-box to the roasting hearth and the fuse-box itself; the former is patched during the run with raw clay mixed with some burned clay; to repair the latter the furnace has to be shut down. Water-cooling of these parts has, as far as the writer is aware, not yet been tried. The following analysis of a torn-out quartz bottom easily explains how the corrosion has taken place: SiO_2 43.6%, FeO 31.5%, MnO 1.0%, Pb 6.5%, Ag 6.1 and Au 0.5 oz. The cost of building a furnace in Pueblo or Denver, as shown in the drawings, is $3,000. The materials required are: Cast iron, 12,000 lb.; wrought iron, 4,000 lb.; sheet iron, 400 lb.; old rail buckstays, 10,000 lb.; red brick, 86,000; firebrick, 15,000.

The tools required by each roaster-man are: Two paddles (blade 5 by 8 in. of $\frac{1}{8}$-in. iron, handle 10 or 12 ft. long of 1-in. iron); two rabbles (head 3 by 9 in. of $\frac{1}{8}$-in. iron, handle 12 ft. long of $\frac{3}{8}$-in. iron); one slice-bar ($1\frac{1}{2}$-in. iron stem flattened to a chisel-point 3 or 4 in. wide), and two door hooks. The front-man has two scoops for the coal, two slice-bars, three rabbles (head 4 by 9 in., of $\frac{1}{2}$-in. iron; handle 10 ft. long of $\frac{7}{8}$-in. iron), and the necessary slag-pots to receive the slagged ore.

The mode of working a furnace with a fuse-box is pretty uniform at the different smelting works. The ore crushed to pass a 2-mesh sieve is sometimes dried on the roof of the furnace, but generally it is dropped directly, without drying, through the hopper into the coolest part of the furnace, the weight of the charge varying from 2,400 to 3,300 lb. according to the thickness of bed the ore can bear. Sometimes the ore is shoveled on to the hearth through the last two doors, but this is only permissible when dropping through a hopper is impracticable. The

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charge is spread uniformly with paddle and rabble to a thickness of about 4 in. over the highest point of the hearth. If this is not separated by steps, but merely inclined, the charge is so spread that it shall lie in front of the first two doors. There it remains until the slagged ore is drawn from the fuse-box, when it is moved down the furnace to its second place on the next hearth, or in front of the next two doors. During its journey to the fuse-box it is not only turned over with the paddle while being moved, but is raked with the rabble once, twice, or three times, according to the interval of time between the movings, *e.g.*, every 1½ hours with 3-hour movings. Before the charge is transferred to the fuse-box the latter receives some silicious ore to protect the bottom. After dropping the charge the fire is urged. The liquefying begins at the surface, and much rabbling is required to bring the unfused parts from the bottom to the top. This is done at intervals of half an hour for 15 minutes at a time at the beginning, later on for 10 minutes, and toward the end for five. Sometimes the fusing ore is only rabbled once, 1½ hours after dropping from the roasting hearth. When the charge is fused it is drawn into slag-pots. This may be done in two ways, either all at once or in three instalments. The former method is used with charges containing 10% lead, the latter with those running high, say 50 and 60% lead, for if they remain any length of time in contact with the acid hearth material they will take up silica and corrode the hearth unnecessarily. As soon as the front-man has drawn the slagged ore from the fuse-box the roaster-men drop in the next charge and begin to transfer the other charges downward, until the hearth near the flue is emptied and ready for a new charge. When the fused ore has become cold it is dumped from the slag-pots, broken up, and transferred to the feed floor of the blast furnace. Its composition must of course vary greatly. As made to-day from mixed sulphide ores, it will average: SiO_2 30 to 40%, Fe 21 to 25%, Zn 9 to 10%, Pb 10 to 15%, S about 2%.

A furnace like the one shown in the drawings roasts and fuses in 24 hours from six to eight charges (varying in weight from 2,400 to 3,300 lb.), consumes from three to four tons of bituminous coal, which is half-lump and half-pea, and requires three men in a 12-hour shift—one front-man, who attends to the fuse-box and firing, and two roaster-men, who do the work on the

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roaster hearth. If the ore is not fused two men are sufficient to do the work. The cost of roasting and fusing with coal at $1.75 per ton, and labor $2.25 for 12 hours, is $2 per ton.

In making up sulphide ore-beds for fusing furnaces two kinds of ore have to be considered: galena concentrates free from or low in silver, assaying 50 to 60% lead, and mixed argentiferous sulphides containing 10% and less lead. With the former class of ore Cramer von Clausbruch* states that at the Altenau smelting and refining works (Harz Mountains) he obtains the best results in treating his galena ores if the charge contains 15% silica and from 55 to 60% lead. If there is more silica a base has to be added to effect a complete slagging at a reasonably low temperature; if there is less, some lead sulphate remains undecomposed. He notes the interesting fact that, if the roasted ore is not completely slagged but retains parts of sulphides and sulphates that have been only agglomerated, the silver and copper will be concentrated in the agglomerated part. One hundred parts of his charge give 85% thoroughly slagged ore, 10% of a mixture of slagged and agglomerated ore, and from 2 to 3% of unroasted agglomerated galena, the loss in roasting varying from 2 to 3%. The slagged part of the charge contains one-half of the silver and only a trace of copper, while the other half of the silver and all the copper are concentrated in the rest of the charge.

At Pribram† the galena concentrate beds contain SiO_2 20%, Pb 35%, and Ag 76 oz. per ton; at Mechernich,‡ SiO_2 22%, Pb 58%, Ag 4 oz. per ton. (Analysis of roasted ore, see p. 171.)

With the latter class of ore the principal base to combine with the silica of the sulphide ore will be iron, and the one next in importance lead. Charges are made up so that they may be readily fusible and sufficiently acid not to corrode the bottom and side walls of the fuse-box. They should contain from 25 to 32% silica, and the iron calculated as metallic iron be made to equal the silica.

Sometimes, however, proportions such as SiO_2 10 to 15%, Fe 20 to 28%, Pb 28 to 20% are formed, owing to the varying amounts of lead sulphide ore and sulphurets on hand, from which

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* *Zeitschrift für Berg-, Hütten-und Salinen-Wesen in Preussen*, xxxi., p. 26; *Engineering and Mining Journal*, March 24, 1883.

† *Oesterreichisches Jahrbuch*, xxxviii, p. 10.

‡ *Berg- und Hüttenmännische Zeitung*, 1875, p. 129.

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the charges have to be made up. It is not common to add lime to a charge containing lead. Sulphide ores free from lead are not often slag-roasted. It may become necessary when the main ores for making up a charge are silicious sulphide concentrates and more or less sandy lead carbonate ores. As some lump ore is necessary to do satisfactory work in the blast furnace and the smelter does not, of course, want to add more slag to his charge than is absolutely necessary, he will slag-roast the pyritic concentrates. As long as these are free from lead the loss in precious metal by volatilization is very low, although the temperature, for the very reason that lead is absent, must be high in order to obtain a satisfactory fusion.

The Long-Bedded Hand Roasting Furnace with Sinter Hearth resembles very much the one with a slagging hearth, the main difference being that, on account of the low temperature required for sintering, the sinter hearth has the same width as the roasting hearth and is separated from the latter by a step, say 8 in. high, and the roof of the sinter hearth has less pitch.

The main dimensions of such a furnace are: Roasting hearth, 74 by 14 ft.; sinter hearth, 10 ft. 4 in. by 14 ft.; grate, 9 ft. 9 in. by 2 ft. 10 in.; firebridge, 9 ft. 9 in. by 1 ft. 6 in.; height of firebridge above grate, 11 in., above sinter hearth 11 in.; height of flue above firebridge, 12 in. at sides and 18 in. at center; height of step separating sinter and roasting hearth, 8 in.; height of flue leading from sinter to roasting hearth, the same as that of roasting hearth, viz., 17 in. at sides and 25½ in. at center. An ore charge, crushed through a 2-mesh sieve and consisting of pyrite with galena and blende, contains, SiO_2 10 to 25%, Fe 20 to 30%, Pb 2 to 15%, Cu 1 to 3%, Zn 2 to 10%, S 28 to 44%, Ag 15 to 35 oz., and Au up to 1 oz. per ton. It remains 30 hours in the furnace, is stirred on the roasting hearth every ¾ hour, on the sinter hearth every ½ hour, drawn every 3 hours, and retains 2 to 3% sulphur. Sintered ore, while it is being drawn from the furnace into a slag-pot, is usually pounded down (say with an iron disk 1 ft. in diameter attached to a handle). Iles says that if a number of holes are punched, with a steel bar for example, into the cake, while still hot, to allow the gases to escape, the center of it when cold will be solid, while if the holes have not been punched through the top it will be loose and coarsely granular.

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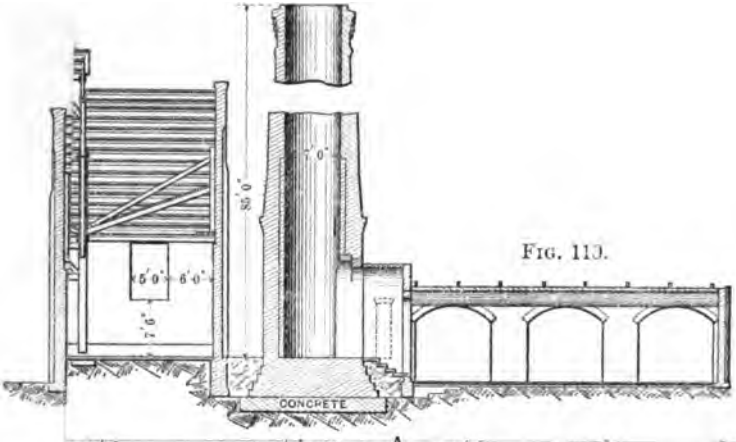


FIG. 113.

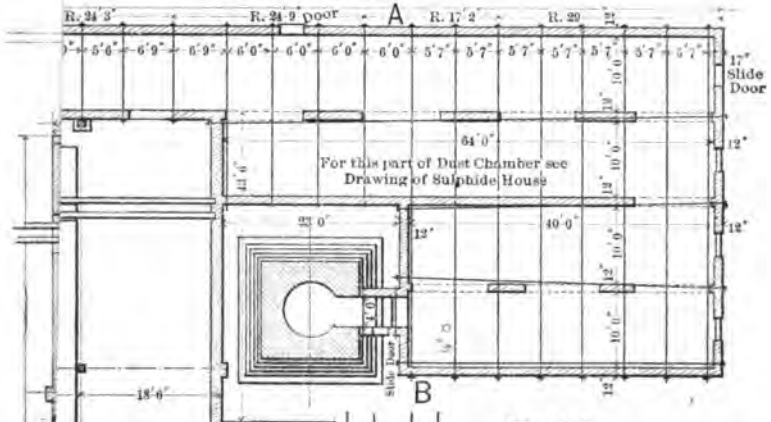


FIG. 111.

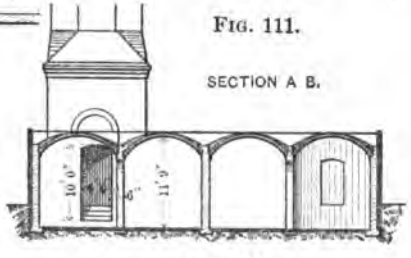
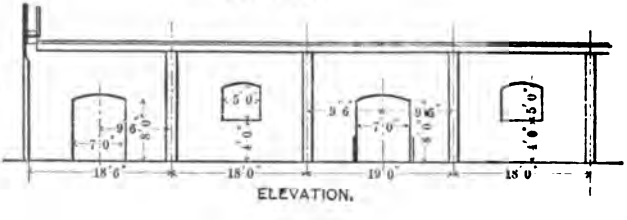


FIG. 112.



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The furnace puts through in 24 hours 12 tons of raw ore, requires three men on a shift, and consumes about three tons of coal.

Figs. 108 to 112 represent a roaster-building with six long-hearth hand-roasting furnaces with slagging hearths. The general arrangement holds good, of course, for any hand reverberatory roasting plant.

Near the flue end of the furnaces, 10 ft. 8 in. above the furnace floor, runs a track, over which pass the ore-buggies to be discharged into the furnaces. At the opposite end of the building is another track at the same elevation for the coal-buggies delivering the necessary fuel into the bins. The flues of the six furnaces lead into a dust-chamber, 10 by 10 ft., up to the spring of the arch. It is 500 ft. long, and is connected through a flue 4 by 10 ft. with a circular brick stack which has an inner diameter of 7 ft. and is 85 ft. high. The dust chamber has sliding doors to discharge the fluedust, two of which are shown in the drawing.

The two products obtained by roasting are roasted ore and fluedust. The composition of some ores running high in lead that have been agglomerated or completely slagged is given below:

	Rodna, Transylvania.		Mechernich, Prussia.		Freiberg, Saxony.	Hall Valley, Colo.	Mine La Motte, Mo.
	Raw Ore.	Sintered Ore.	Raw Ore.	Slag-roasted Ore.	Slag-roasted Ore.	Roasted Ore.	Sintered Ore.
Pb	47.29	54.27	60.40	62.08	.....	.....	75.95
PbO	.....	.....	.....	.....	22.0	42.04	.....
Ag	0.059	0.061	0.0105	.....	0.13	Ag_2O,0.21	.....
Au	0.0001	0.0001	.....	.....	.....	.....	Ni-Co, 1.11
Cu	Trace.	0.02	0.17	0.14	.....	.....	.....
Cu_2O	.....	.....	.....	.....	0.8	.....	.....
CuO	.....	.....	.....	.....	.....	1.71	.....
As	0.34	0.030	NiO.10	n. d.	.....	.....	.....
As_2O_3	.....	.....	.....	.....	1.1	.....	.....
Sb	0.02	0.027	0.07	0.08	.....	.....	.....
Fe	20.36	24.06	0.80	0.56	.....	.....	.....
FeO	.....	.....	.....	.....	.....	3.59	4.5
Fe_2O_3	.....	.....	.....	.....	33.3	.....	.....
Zn	0.67	0.87	0.15	n. d.	16.0	.....	.....
ZnO	.....	.....	.....	.....	.....	.....	.....
Al_2O_3	0.11	0.23	3.60	4.24	1.8	8.11	.....
CaO	Trace.	Trace.	0.88	1.28	2.0	0.42	0.76
MgO	Trace.	Trace.	.....	.....	0.5	.....	.....
BaO	.....	.....	.....	.....	.....	12.05	.....
SiO_2	0.49	0.80	22.05	27.77	17.4	22.71	7.21
CO_2	.....	.....	.....	.....	.....	.....	.....
SO_3	.....	2.25	.....	.....	Trace.	6.31	.....
S	29.86	2.72	9.72	0.60	3.6	2.94	7.41
O	.....	13.41	.....	.....	.....	.....	.....
Reference:	(a)	(a)	(b)	(b)	(c)	(d)	(e)

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	Omaha & Grant, Denver, Colo.		Globe Works, Denver, Colo.				Leadville, Colo.		
	Slag-roasted Ore.		Sintered Ore.				Fused Pyritic Ore.		Roasted Ore.
Pb	13-15	13.15	20.5	21.8	15.0	27.0	.....	.....	.....
PbO	.....	.....	.....	.....	.....	.....	.....	.....	27.06
Ag, oz	.....	.....	28.0	91.5	57.0	75.0	28.0	30.0	42.75
Au, oz	.....	.....	0.26	0.24	0.51	0.24	0.13	0.12	.....
Cu	.....	.....	.....	.....	.....	.....	.....	.....	.....
Cu_2O	.....	.....	.....	.....	.....	.....	.....	.....	.....
CuO	.....	.....	.....	.....	.....	.....	.....	.....	0.66
As	.....	.....	.....	.....	.....	.....	.....	.....	.....
As_2O_5	.....	.....	.....	.....	.....	.....	.....	.....	Mn_2O_4
Sb	.....	.....	.....	.....	.....	.....	.....	.....	0.23
Fe	25.3	22.7	30.8	21.8	32.6	29.2	.....	.....	.....
FeO	.....	.....	.....	.....	.....	.....	53.1	74.8	.....
Fe_2O_3	.....	.....	.....	.....	.....	.....	.....	.....	32.69
Zn	10.5	9.5	4.6	7.3	4.0	4.1	.....	.....	.....
ZnO	.....	.....	.....	.....	.....	.....	.....	.....	30.32
Al_2O_3	.....	.....	.....	.....	.....	.....	.....	.....	2.31
CaO	.....	.....	.....	.....	.....	.....	.....	.....	1.36
MgO	.....	.....	.....	.....	.....	.....	.....	.....	.....
BaO	.....	.....	.....	.....	.....	.....	.....	.....	.....
SiO_2	34.7	35.7	15.0	22.0	18.4	10.4	4.7	4.0	4.64
CO_2	.....	.....	.....	.....	.....	.....	.....	.....	.....
SO_3	.....	.....	.....	.....	.....	.....	.....	.....	4.05
S	.....	.....	3.8	4.0	2.8	2.1	.....	6.0	1.00
O	.....	.....	.....	.....	.....	.....	.....	.....	.....
Ref	(*f*)	(*f*)	(*g*)	(*g*)	(*g*)	(*g*)	(*h*)	(*h*)	(*i*)

(*a*) *Oesterreichisches Jahrbuch*, xxix., p. 27. (*b*) *Berg- und Hüttenmännische Zeitung*, 1875, p. 129. (*c*) *Oesterreichisches Jahrbuch*, xvi., p. 397. (*d*) "Transactions of American Institute of Mining Engineers," v., p. 568. (*e*) *School of Mines Quarterly*, ix., p. 216. (*f*) Livingstone, private communication, August, 1896. (*g*) Iles, private communication, August, 1896. (*h*) Private notes. (*i*) Dewey, Bulletin 42, U. S. National Museum, p. 48.

An analysis of crystals of slag-roasted galena ore rich in zinc gave, according to Heberdey,* SiO_2 16.62%, PbO 61.50%, ZnO 18.26%, FeO 1.69%, CaO trace, MgO 1.99%; sp. gr. 5.214.

The amount of fluedust carried off with the gases is about 2%, and is practically all collected in the dust chambers; of the metal volatilized in fusing very little if any is recovered. Fluedust from furnaces when the ore is simply roasted has a brownish color; if slagging is carried on, it is gray from volatilized lead and zinc. This gray dust, if it contains not less than from 1 to 2% zinc has, when moistened, the property of solidifying to a hard mass. This is not the case with brown fluedust. The solidifying is probably caused by the anhydrous zinc sulphate becoming hydrated. Its binding property is strong enough to allow the mixing in of 25% of non-binding material, when it still will form a hard brick when molded. In fact, the gray dust, when moistened, forms a pasty mass requiring a stiffening ingredient to

* Kroupa, *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1893, p. 138.

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permit its being molded at all. The composition of some samples of fluedust is subjoined.

	Friedrichshütte, Silesia. (a)	Freiberg, Saxony. (b)			Přibram, Bohemia. (c)	Omaha & Grant Works, (d) Denv'r, Col.	Globe Works, Denver, Colo. (e)		Globe Works, Denver, Colo. (f)	
Pb	.....	26.27	21.37	16.27	.....	.....	18.70	12.60	5-10	19-40
PbO	62.80	.....	.....	.....	45.50	.....	.....	.....	.....	.....
$PbSO_4$	.....	.....	.....	.....	.....	57.23	.....	.....	.....	.....
Ag, oz.	23.33	.....	.....	.....	46.66	.....	37.70	46.60	23-23	20-37
Au, oz.	.....	.....	.....	.....	.....	.....	0.15	0.26	0.50-1.50	0.10-0.75
Cu	.....	.....	.....	0.02	Trace.	.....	2.70	2.70	.....	.....
Cl	.....	.....	.....	.....	Trace.	.....	.....	.....	.....	.....
Sb_2O_3	.....	.....	.....	.....	1.35	.....	.....	.....	.....	.....
As	.....	7.56	37.5	46.41	.....	.....	0.30	0.30	.....	.....
As_2O_3	.....	.....	.....	.....	1.06	15.86	.....	.....	.....	.....
Zn	.....	19.10	2.11	0.45	.....	.....	6.30	6.10	None.	7-8
ZnO	3.20	.....	.....	.....	2.06	1.50	.....	.....	.....	.....
Fe	.....	.....	.....	.....	.....	.....	13.80	.....	.....	.....
FeO	.....	.....	.....	.....	.....	1.30	.....	.....	.....	.....
Fe_2O_3	4.44	.....	4.36	1.68	.....	.....	.....	.....	.....	.....
Fe_3O_4	.....	.....	.....	.....	7.20	.....	.....	.....	.....	.....
MnO	.....	.....	.....	.....	1.14	.....	.....	.....	.....	.....
Al_2O_3	.....	.....	.....	.....	0.53	.....	.....	.....	.....	.....
CaO	1.96	0.80	0.61	0.43	0.45	.....	.....	.....	.....	.....
MnO	.....	0.50	0.25	0.15	.....	.....	.....	.....	.....	.....
SiO_2	.....	8.90	4.81	2.53	(=10.0)	.....	16.10	.....	.....	.....
Insol.	.....	.....	.....	.....	14.20	.....	.....	.....	.....	.....
SO_3	25.80	28.14	8.23	7.95	.....	.....	.....	18.20	.....	.....
H_2SO_4	.....	.....	.....	.....	23.28	17.13	.....	.....	.....	.....
S	.....	.....	1.34	0.60	Trace.	.....	8.20	8.60	.....	.....
C	.....	.....	3.40	3.75	2.50	.....	.....	.....	.....	.....
H_2O	1.50	.....	1.46	3.27	.....	.....	(g)	(h)	(i)	(j)

(a) Kosmann, *Zeitschrift für Berg-, Hütten- und Salinen- Wesen in Preussen*, XXXI., p. 227. (b) Hering, "Die Verdichtung des Hüttenrauches," Stuttgart, 1868, p. 34. (c) Mann, *Oesterreichisches Jahrbuch*, 1890, XXXIX., p. 15. (d) Livingstone, private notes, 1896. (e) Iles, private notes, 1896. (f) Iles, *Engineering and Mining Journal*, Jan. 30, 1896. (g) Average of 10 years. (h) From fusing furnaces. (i) From sintering furnaces. (j) From roasting furnaces.

The large quantity of gold shown in some of the analyses is probably due to gold-bearing pyrite concentrates from gold mills used as iron flux, and to telluride gold ores. The manner of working fluedust is discussed under the head of "Fluedust from Blast Furnaces," § 94.

§ 59. THE ROPP STRAIGHT-LINE FURNACE.—This furnace, represented in Figs. 113 to 117, is a single-hearth mechanical roasting furnace, 150 ft. long by 14 ft. wide, with four exterior fireplaces *b*. The hearth *a* has the form of the ordinary hand reverberatory roasting furnace. Along its center runs a straight narrow channel *c*, 1½ in. wide, through which extend at equal distances six pairs of vertical arms *d*, attached to 12 four-wheeled trucks *e*, and carrying each a rake of the width of the hearth, the blades of which *g* are placed at an angle of 45°. The rakes,

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on entering near the flue, at *h*, where the ore is fed automatically by a pair of Hendy Challenge Ore Feeders *i*, carry it slowly to the discharge *j*, where it falls into trucks *k*. Inlet and outlet are closed by swinging doors. On leaving the furnace the rakes return again on an outside rail *m* and re-enter at the same place as before. Below the inside rails *n*, over which pass the carriages, is an underground passage *o* deep enough for a man to stand in and attend to any necessary repairs. A steel rope *p*, $\frac{5}{8}$ in. in diameter, connecting the carriages, passes on entering and leaving the furnace over horizontal sheaves *q*, the first of which is connected with the power. The vertical shaft *r* of this sheave is connected by means of bevel gearing *s* and *s'*, with a horizontal shaft *t*, which receives its power through spur-wheels *u* and *u*, from a second shaft driven by a pulley *v*. The 75 revolutions per minute of the latter are converted by two reductions into 1.63 per minute of the sheaves in the 14-ft. furnace. The steel rope, which does not become hotter than the hand can bear, lasts about $1\frac{1}{2}$ years and the rakes are thoroughly cooled on their return trip. At the date of writing (April, 1897) they have lasted 14 months and are estimated to be good for six months more. The furnace, as will be seen, is very simple in construction. It has working doors on the sides which permit access to the hearth, and air is admitted as freely as in the hand reverberatory furnace. The heat is regulated from the different fireplaces, which have each a perforated firebridge to insure complete combustion with air superheated in the side walls; the products of combustion pass off through the smoke flue *w*. The longitudinal tie-rods, it will be noticed (Fig. 113), do not exceed 50 ft. in length, $5\frac{1}{2}$ in. space being left in the side walls for buckstays.

The weight of the iron work of the furnace exclusive of buckstays is about 76,000 lb.; the addition of the buckstays weighing about 18,000 lb. makes a total of about 94,000 lb. About 164,000 red brick and 29,000 firebrick are required. The cost of the iron work f. o. b. San Francisco (Parke & Lacy Co., manufacturers) is, including buckstays, $6,500; exclusive of buckstays, $6,250. This includes steel rope, two feeders, and the royalty.

The furnace is built also in two smaller sizes, 100 by 14 ft. and 105 by 11 ft. Some of the work done by the furnace is shown in the subjoined table.

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THE ROPP STRAIGHT-LINE FURNACE.

Length of hearth		105′	100′	150′
Width of hearth		11′	14′	14′
Hearth area (deducting slot)		1,120	1,307	2,050
Number of fireplaces		3	3	4
Length of single grate		4′ 6″	5′	5′
Width of single grate		4′	4′	4′
Grate area, square feet		18	20	20
Ratio hearth to grate area		64 : 1	70 : 1	105 : 1
Number of carriages		4	4–6	6
Number of blades on carriage		26	32	32
Carriage makes the circuit in (minutes)		3½	3½	5½
Ore stirred every (seconds)		52	53	53
Horse-power required		4–5	5	6–8
Depth of charge near fluebridge	Pyrite	1″–2½″	Same.	Same.
	Matte	1′ 1–3″		
Time ore remains in furnace (hours)		6–8	6–8	6–8
Tons in 24 hours	Pyrite	26–42	35–50	45–70
	Matte	26–34	32–45	45–60
	Dry gold ore			80–98
Per cent. sulphur in pulverulent roasted	Pyrite	0.1–8	Same.	Same.
	Matte	1–8		
	Dry gold ore	0.14–0.16		
Number of men in 12-hour shift		1–3	Same.	Same.
Tons of coal per ton of ore		1/8–1/15	1/10–1/14	1/11–1/18
Gallons lubricating oil per ton of ore		15	14	18

Character, composition and size of ore.
Pyrite: SiO_2 18–30, Fe 20–30, Cu 0–6, Pb 0–20, S 20–30, Zn 0–10; 30-mesh to ¼-in.
Matte: Fe 20–40, Cu 0–50, Pb 0–18, S 18–30, Zn 0–4; 20-mesh to ½-in.
Dry Gold Ore: S 1–2, 30-mesh and finer.

§ 60. THE PEARCE TURRET FURNACE.—The Pearce furnace is a mechanical roaster the hearth of which is circular or turret-shaped, with an open space left for the discharge of the ore. It is heated from external fireplaces. The ore fed automatically through the roof at one side of the open space is turned over and moved along the hearth to the discharge on the opposite side by rabble blades attached to a pair of horizontal pipe arms inserted radially in a hollow hub revolving around a stationary cast-iron column. Part of the air necessary for oxidation is forced through the central column into the rabble arms, being thus warmed and protecting the iron work. The inner wall of the furnace having for the passage of the rabble arms a continuous slot, made air-tight by a traveling steel tape, has its upper part above the slot suspended from I-beams resting on the outer furnace-wall and the central column.

The furnace, as built with one hearth, is shown in Figs. 118 and 119; and with two hearths, one above the other, in Figs. 120 and 121. The details of the rabble blades, and of closing the rabble-arm slot, are given in Figs. 122 to 125.* The outer diame-

* Peters, "Modern Copper Smelting," 1895, p. 207.

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ter of the furnace with a 6-ft. hearth is 36 ft., the inner diameter 19 ft. 4 in.; the fireplaces project 6 ft.—thus a floor space of 36 by 42 ft., or 1,512 sq. ft., is required. The height of the single-hearth furnace is 11 ft.; of the double-hearth furnace, 16 ft. 6 in. The furnaces are tied on the outside by wrought-iron bands and braced on the inside by radial struts.

The single-hearth furnace has undergone some changes in the last few years, and Figs. 118 and 119 represent its latest form. The hearth *a* is usually 6 ft. ½ in. wide in the clear, having an area of 505 sq. ft.; some furnaces have been built 7 ft. (area 609 sq. ft.) and 8 ft. (area 788 sq. ft.) wide without affecting the quality of the work. Under the hearth there used to be a dust chamber *b* (as is still the case with the two-hearth furnace, Fig. 121) into which the gases passed from the downtake *c* (Fig. 118). This has been given up, the space under the hearth being filled up solid, thus cheapening the erection of the furnace.

The outer wall (Fig. 120) has a number of working doors *d* permitting access to the hearth; between each pair of doors and 2½ in. above the level of the hearth (Figs. 119 and 120) is a horizontal flue *e*, through which air, warmed in its passage, can be admitted at the surface of the roasting ore, provided the draft in the furnace is sufficiently strong. When the fireplaces are worked with under-grate blast, or when oil is used as fuel, very little air enters through these openings. The inner wall is divided by the rabble-arm slot *f* (Fig. 119) into two parts, the lower one erected in the usual way and the upper one suspended (Figs. 118 and 119) by means of stirrups *g* from I-beams *h* and cross-beams *i*, and braced by radial struts *c'*, and angle irons *c''*. The I-beams are supported by the outer wall and the central column *j*, and the cross-beams by the I-beams.

The three external fireplaces *k*, with their coal hoppers *l*, are shown in Figs. 118, 120 and 121. The flame from the fireplace nearest the ore-feed passes over a curtain arch *m''* which prevents the raw ore from being overheated. Fig. 121 represents a furnace with a step-grate for slack coal and lignite. With lump coal flat grates are used, and oil has in many instances replaced solid fuel. The stirring mechanism, which at first sight appears complicated, is nevertheless simple. The rabbles *m*, of ¼-in. steel plate, are (Figs. 118, 119, 122, 123) graduated in length and direction from the inner to the outer circle, so that the ore

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FIG. 119.

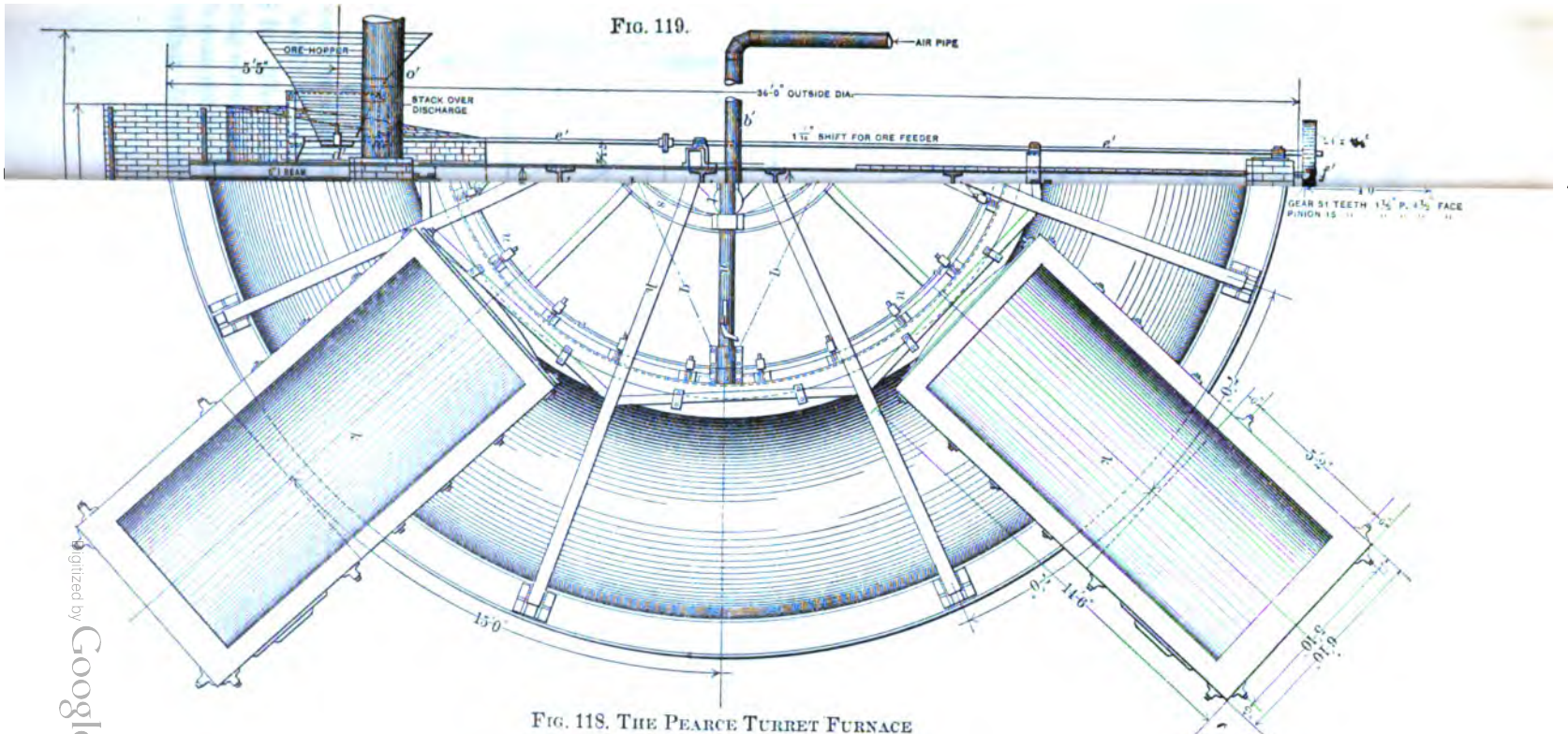
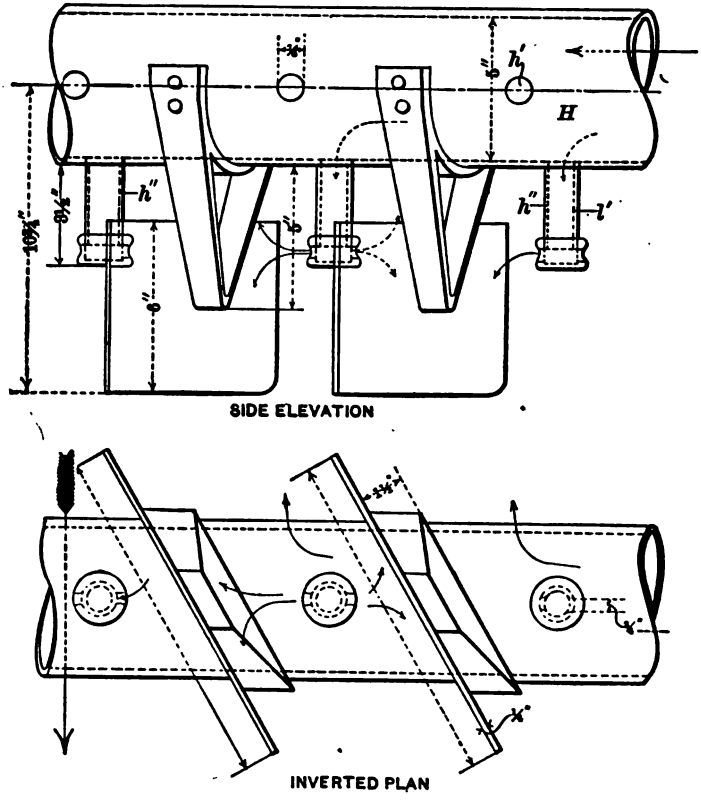


FIG. 118. THE PEARCE TURRET FURNACE

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on the outer circle, which has to travel a greater distance, may remain in line with that near the inner circle. In fact, the placing of the rabbles has to be absolutely correct, if the ore is not to accumulate on one of the sides and thus cause imperfect work. It used to be supposed that, in order to obtain an even roast, more ore must be fed at the outer than at the inner circle, as the

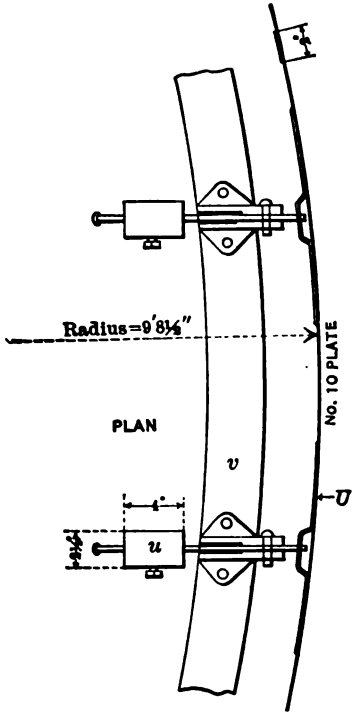


FIGS. 122 AND 123.—PEARCE TURRET FURNACE—DETAIL OF RABBLES.

former was stirred more than the latter. If this had been true, the width of the furnaces would not have been increased from 6 to 8 ft. The rabbles last from 4 to 6 weeks with pyritic ore containing about 40% sulphur. The furnace was at first provided with four rabble arms *n* (Figs. 118 and 119), placed at 90° and radiating from the hub *o*, which revolves around the hollow

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cast-iron column *j*. To-day only two arms, diagonally opposite each other, reach into the furnace, while the other two form the reserve; the four arms are braced by rods *q*. A rabble arm



FIGS. 124 AND 125.—PEARCE TURRET FURNACE.—DETAIL OF CLOSING RABBLE-ARM SLOT.

consists of two pieces of 5-in. gas-pipe joined at *r*. This permits its being readjusted in the furnace to the wearing-off of the rabbles and exchanged through one of the doors in the outer wall

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(see *d*, Fig. 120) for another having new blades, an operation which lasts only a few minutes. The air passing through the rabble arms enters the furnace (Figs. 122 and 123) through horizontal holes *h'* and small pipes *h''*, pointing downward. To the rabble arms is attached a spur wheel *s* (centered by rollers *t*) in gear with pinions *c'* to the pinion shafts *v*, being driven by bevel gearing *w* and the driving shaft *x*. In the earlier furnaces the spur wheel *s* was driven, as shown in Fig. 121, by a single pinion, and its shaft when passing through the dust-chamber was inclosed in a 5-in. wrought-iron pipe. The I-beams *y* serve as inner braces of the furnaces; they hold the boxes of the vertical pinion shafts and carry the circular track *z*, on which run the conical rollers *a'*, carrying the entire weight of the rabble arms and driving gear. The blast enters column *j* through the pipe *b'*, passes into the hub *o*, acting as a wind-box, and thence into the arms. The outer part of the rabble arm exposed to the heat lasts about one year. The rabble arm slot *f* is closed by pieces of 12-in. steel tape (Figs. 124 and 125) carried by the rabble arms and pressed against the slot by weighted bell cranks *u*, attached to the circular angle iron *v*. The arms travel at an average speed of 75 ft. per minute, taking the circumference of the center of the hearth as the line of measurement. This makes with a 6-ft. hearth the time of a complete circuit 53 seconds, and causes the ore to be stirred once in 26½ seconds.

The ore, which remains about 6 hours in the furnace, is fed automatically on to the hearth. This was formerly accomplished by the rabble arms opening the discharge of the ore-hopper *d'* when passing under it. To-day the feeder under the ore-hopper is controlled through a separate shaft *e'*, connected by pulleys *f'* and *f''* and belting *g'* with the main driving shaft *x*; thus the feeding is regulated by the speed of the rabble arms, but does not jar them as they pass over the hearth.

A rabble arm after leaving the discharge opening travels over the open space and arrives at the uncovered part of the hearth which receives the ore from the feeder. It spreads this ore out and enters the covered hearth by the swinging door *i'*. As at first the raw ore is only to be heated up to the point where it begins to roast, any air passing through the arms would retard the process. It is therefore automatically controlled by a butterfly valve *j'* in the arm, which closes as it strikes the first

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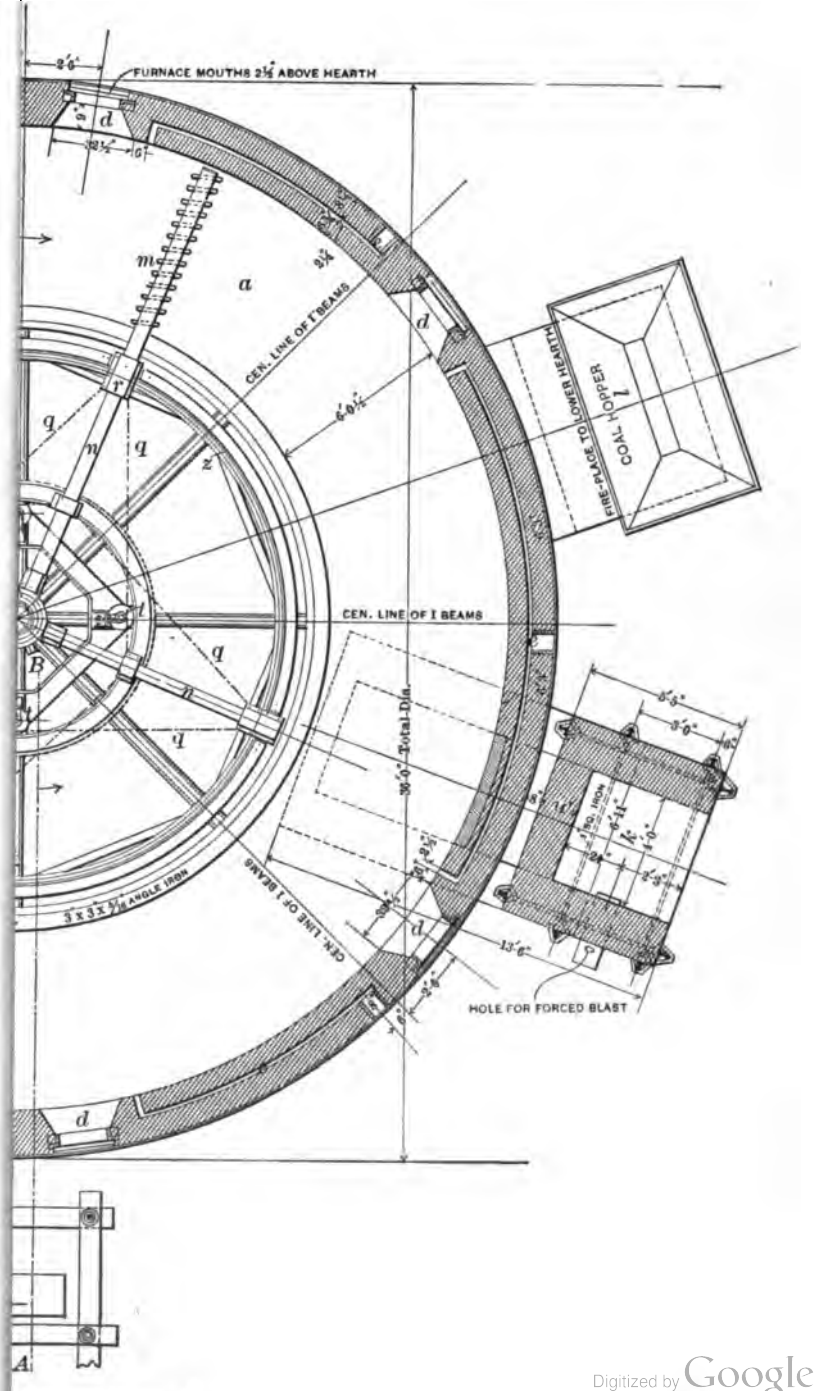
THE PEARCE SINGLE HEARTH FURNACE.

Hearth.		Roasting Material.			Production in 24 Hours.	Sulphur in Roasted Material.	Material Roasted per Square Foot of Hearth Area.	Hearth to Grate Area.	Coal Consumed per Ton of Ore.	Ore Remains in Furnace.	Rabble Arm Makes One Circuit in Seconds.	Authority.
Width. Feet.	Area. Square Feet.	Kind.	Size Mesh.	Composition.	Tons.	Per Cent.	Pounds.	Ratio.	Pounds.	Hours.		
(a) 7	609	Pyrite.	.....	SiO_2 25–30, S 35.	22	3	72	15:1	318	.....	70	Pearce.
(a) 7	609	Matte.	6	Cu 40-50, Pb up to 15	12	3	40	15:1	583	.....	70	Pearce.
6	505	Pyrite.	2	S 43	16	6-7	63	.....	328	6	80	Peters.
6	505	Matte.	6	Cu 15, Pb 11, S 17.	11	3.3	40	.....	.....	6	80	Peters.
6	505	Pyrite.	.....	SiO_2 10, S 45.	9	None.	35	.....	.....	6	80	Peters.
6	505	Mixed Sulphide.	.....	SiO_2 18, Fe 20, Zn 15, Cu 11, S 31.	15	6-7	60	.....	.....	6	80	Peters.
6	505	Mixed Sulphide.	30-2	SiO_2 18–30, Fe 20–30, Pb 0–20, Zn 0–8, Cu 0, S 25–30.	14-16	1-8	60	.....	333	4-5	104	Private notes.
6	505	Matte.	20-2	Fe 20–40, Pb 0–18, Zn 0–15, Cu 0–50, S 18–30.	10-14	1-7	47	.....	333	4-5	104	Private notes.
6	505	Silicious Gold Ore.	12	S 1-3	50-60	0.2	210	(b)	(b)	.....	70	Stearns-Rogers Mfg. Co.
6	505	Matte.	8	Fe 55, Pb 12, S 29.	30	5-6	118	17:1	333	6	80	Setz.

(a) Three fireplaces, each 4 ft. 6 in. by 8 ft.

(b) Oil burners installed in crown of arch.

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TURRET FURNACE

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stop k' and opens again as it strikes the second stop m'. The arm leaves the covered hearth by a second swinging door i'', carrying with it some roasted ore which drops through the discharge-hopper n' into a wheelbarrow or truck, the fumes being carried off by a stack o' made of thin sheet iron.

For a furnace with a 6-ft. hearth are required: Iron work, 40,000 lb.; red brick, 65,000; firebrick, 3,500. A furnace of this kind when completed costs in Denver, Colo., including royalty, $5,500. For a 7-ft. hearth the materials required are: Iron work, 50,000 lb.; red brick, 90,000; firebrick, 4,000.

Some of the work done by the single-hearth furnace is shown in the preceding table.

The two-hearth furnace, occupying the same floor space as the single-hearth furnace, furnishes twice the hearth area, or 110 sq. ft. with a 6-ft. hearth. The details of the upper hearth a' are nearly the same as those of the lower hearth, and in Figs. 120 and 121 similar parts have been designated with the same letters. Of special interest is the manner of supporting the inner wall of the upper hearth. Its lower part is supported by I-beams h, while the upper is suspended from the top I-beams h''' in the same manner as the roof of the single hearth furnace in Fig. 119. The upper hearth has from two to three fireplaces, the lower hearth usually three. The furnace represented in Figs. 120 and 121 has two upper fireplaces k' and two lower ones k. Each hearth has two rabble arms. The ore, fed as with the single-hearth furnace, travels in the direction of the arrow over the upper hearth, drops through a slot p' to the lower hearth and moves in the same direction to the discharge. About 5 horse power are required to run the furnace and blower, and one man can attend two furnaces. Some of the work done at the works of the Colorado Smelting and Mining Co., Butte, Mont., reported by Mr. H. Williams, is subjoined. The furnace is 7 ft. wide (hearth area 1,218 sq. ft.) and has two fireplaces for each hearth (one 4 ft. by 2 ft. 10 in., the other three 3 ft. by 2 ft. 10 in., grate area, 236.8 sq. ft.; ratio hearth area to grate area as 33:1). There are four rabble arms, each having 14 rabbles. An arm making the circuit in 45 seconds; the ore is therefore stirred every 22½ seconds. An average sample of raw ore contains SiO_2 16%, Cu 12%, Zn 8%, S 35%, Fe 25%, As 3%, and Ag 20 oz. It is crushed through a 4-mesh sieve, and

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lies about 3 in. deep on the hearth near the fluebridge; 24 tons are roasted in 24 hours (39 lb. per sq. ft. of hearth area), the sulphur being reduced to 6%, of which 4% are present as sulphide and 2% as sulphate. The amount of fluedust is 1.2% of the ore charged. Per ton of ore 316 lb. of coal are consumed and 0.013 gal. oil for lubricating. One man attends to each furnace on a shift.

§ 61. The Brown Horseshoe Furnace.—This furnace is ring-shaped, occupying five-sixths of the annular space, the other sixth being an open span for the feed and discharge of the ore and for the cooling of the stirrers. It is heated from three exterior fireplaces. Characteristic of this as of all Brown's roasting furnaces* is that the hearth is inclosed by slotted walls, between which and the outer ones run passages where the carriages travel which bear the stirrers. Thus the mechanism is protected to a considerable extent from the heat and the gases. The carriages are automatically attached to and detached from a cable running around horizontal sheaves placed in the inner passage. Doors, both on the inner and the outer sides, give access to the sheaves, the slotted walls, and the hearth.

The furnace is represented by Figs. 126 to 133. With a hearth 8 ft. wide and 135 ft. long (having an area of 1,080 sq. ft.) the inner diameter is 41 ft. 10 in., the outer 68 ft. 2 in.; the fireplaces project 5 ft. 2 in.; thus a floor space of 6,162 sq. ft. is required. The total height of the furnace is 6 ft.

The plan (Fig. 126) shows the general construction of the furnace in its latest form. In the center is an engine *a* of from 5 to 8 horse-power (although only 2 are required) which drives the mechanical feeder *b* and the cable *c* running over sheaves *d*. Opposite each sheave is a door *e* which gives access to the sheave for oiling and repairing and admits air for cooling the bearings. The outside doors *f* are just far enough apart (4 ft.) to permit access to the hearth when the outer wall *g* between the doors has been removed and the tiling *h* taken out. The flues from the exterior fireplaces are turned in the direction of the draft, so that the flame may spread uniformly under the roof and not strike the ore. Stack *i* carries off the gases and fumes. The ore is fed at *b* by an automatic feeder which delivers with every passage of a carriage a weighed amount on the hearth outside of the

* *Engineering and Mining Journal*, July 4, 1896.

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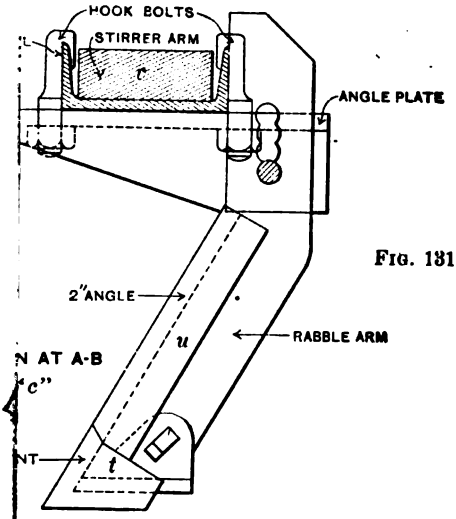


FIG. 131.

DETAILS OF RABBLE ARM.

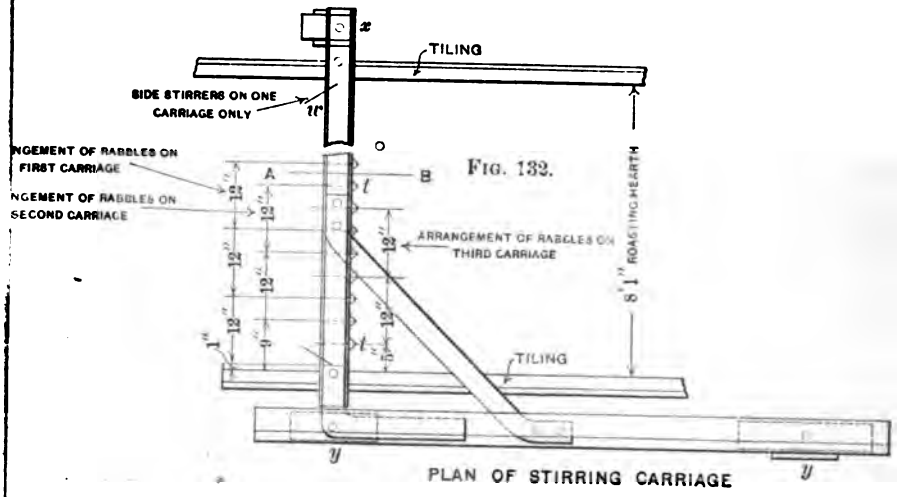
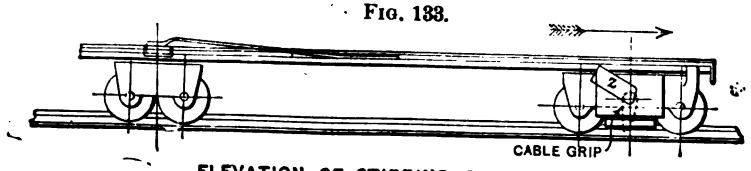


FIG. 132.

PLAN OF STIRRING CARRIAGE

FIG. 133.



ELEVATION OF STIRRING CARRIAGE

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swinging door. The ore enters the furnace through the swinging door *j'*, travels over the hearth, leaves it through the other door *k*, and is discharged into wheelbarrows or trucks placed at *l*. The stirring carriage *m* after leaving the furnace strikes a second one at rest (not shown) in the open space and pushes it along until its grip-lever comes in contact with a fixed stop and is forced down, when it grips the cable which draws the carriage into the furnace. At the same time the heated carriage is automatically released, comes to a stop and cools in the open space for from one to three minutes, the time required for a carriage to make the circuit. In this manner the two carriages are used alternately at intervals of from one to three minutes and are not sufficiently heated to be much affected by their passage through the furnace. The vertical section through the hearth (Fig. 127) gives some details of construction. On the side walls, built to a level with the hearth, are placed the heavy cast-iron frames of the inner and outer doors *e* and *f*, on the frames rest 9-in. channel-beams *n*, secured by the buckstays *o* and the tie-rods *p*. Thus the roof *q* is supported independently by the brickwork between the doors, which can be taken out if necessary. The sheaves *d* are seen to be supported in the doors outside the inner passage, which prevents them from being heated. The tiling *h*, forming the lower part of the slotted wall, is set in the brickwork of the hearth and can be readily removed, should it be necessary to exchange it or gain access to the hearth, without disturbing the roof or cooling down the furnace. The tile *s* forming the upper part of the wall is built into the roof and special precautions must be taken to secure it firmly. The inner passage shows a T-rail for the inner double-flanged wheels of the stirring carriage, while in the outer passage is a smooth rail for the outer flat-tread wheels, giving room for expansion and contraction. At some works the furnace is inclosed by ribbed cast-iron skewbacks at the level of the tracks (not shown) in order to keep these rigidly in place and thus prevent any mishaps to the stirring carriage. Any expansion of the furnace will take place below the track-line where it will do no harm. Figs. 128 to 130 show the details of the fire-boxes which are 3 by 5 ft., making the ratio of hearth to grate area as 24 : 1. The first fire-box is placed about 60 ft. away from the feed end of the furnace, the other two are 35 and 40 ft. apart. A heavy sulphide ore ignites

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at a distance of about 15 ft. from the flue-end or in about 1 hour after it has been fed; an ore running low in sulphur will be heated to the oxidizing temperature in from 2 to 2½ hours. In Figs. 131 to 133 are given the details of the stirring carriage with improved form of stirring plows. The furnace, when first used, had the ordinary rabbles, oblong plates of steel placed obliquely to the arm reaching across the hearth. Such a blade wears from 1¼ to 1½ in. in about a fortnight, when it is turned, and after another two weeks it goes to the scrap heap. The disadvantage of these rabbles is that the lower front point becomes rounded off when the ore is not uniformly stirred; the uneven wear has been diminished by riveting to the steel rabble a white-iron shoe which wears very slowly. When crusts form on the hearth, they are sometimes plowed up, say once in five days, by a curved steel bar attached to a reserve carriage. The bar is shifted about 5 in. after each circuit and the crust broken up in about an hour. Care must, of course, be taken not to allow the crust to become too thick, as, when this is the case, plowing up strains the machinery and becomes almost impossible. Another way is to have a separate frame with steel teeth 1 in. square reaching beneath the ore and run it all the time, thus preventing any crust at all from forming. A third method of working is shown with the Brown-O'Hara furnace.

The improved rabble *t* (Fig. 131) is plow-shaped and of cast iron; it wears on the bottom, but the cutting edge always remains pointed. The plow is keyed to a cast-steel rabble-arm *u* attached to the stirrer arm *v* reaching across the hearth. The arm can be lowered as the plow wears off. When a crust forms the arms are lowered one at a time so as to cut just under it into the loose material placed on the brick hearth. The rabble arms are placed about 12 in. apart on each carriage (Fig. 132) and are mismatched so that with three carriages the whole hearth surface is covered. One of the carriages has side stirrers *w* to prevent any ore from accumulating there. A carriage (Figs. 132 and 133) consists of an L-shaped frame with one flat-tread wheel *x* running in the outer and two double-flanged wheels *y* in the inner passage. The front inner wheel has the cable-grip *z*.

The life of a cable is six months and less according to the strength of the splice and the strain it has to bear. The "Flattened Strand Rope" lately introduced, presenting a much greater

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THE BROWN HORSESHOE FURNACE.

Character of Ore.	Raw Ores.				Ore Roasted in 24 Hours.	Fuel.			Ratio Hearth to Grate.	Ore Roasted per Square Foot Hearth Area.	Roasted Ore, Sulphur.
	Size. Mesh.	Sulphur. Per Cent.	Zinc. Per Cent.	Lead. Per Cent.	Tons.	Wood. Cords.	Coal. Tons.	Per Cent. on Ore.	Area.	Pounds.	Per Cent.
Pyrite and chalcopyrite....	4	40–45	3–5	None.	35	3½	.....	1 cord: 10 tons ore.	24 : 1	48.8	4.5
Galena and blende.....	4	20–25	8	15	25	.....	(a) 4	16	23.3 : 1	47.6	3.5–4
Copper matte.....	40	20	40 Cu	15	20	.....	3.25	16	35 : 1	38.0	(b) 6.0
Silicious ore with pyrite....	30	2.5	1.5 As	.....	95	6	.....	1 cord: 15.83 tons ore.	33 : 1	131.0	0.3
Silicious ore with pyrite....	20	2–3	2.5 As	.....	65	6	.....	1 cord: 10.85 tons ore.	33 : 1	90.0	0.5
Silicious ore with pyrite....	16	2–5	.....	.....	50	.....	(a) 4.5	8.5	28.66 : 1	77.0	0.5
Silicious slimes.....	60–400	2–5	.....	.....	65	.....	5.25	8.0	31.1 : 1	92.8	0.3
Blende.....	12	28	56	1.5	12	.....	(c) 8.0	66	22.8 : 1	22.0	0.8

(a) Lignite. (b) Present as sulphate, the aim of the roast being to peroxidize the iron and convert as much copper as possible into soluble sulphate.
(c) Refuse slack coal.

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wearing surface, is an improvement on the common round cable. The wheels of the carriage, formerly of cast iron and wearing out in two months, are now made of cast steel and last about four months. The cast-iron plows last from four to five times as long as the steel plate.

The materials required for the furnace are: Iron work, 35,000 lb; buckstays, 20,000 lb.; red brick, 60,000; square firebrick, 4,500; wedge-brick and fireclay tiling, 12½ tons. The weight of a sheet metal stack 42 in. by 80 ft., including base plates, is 5,200 lb. The cost of the iron work in Chicago is $3,500, and the cost of the furnace about $7,000. Some of the work done with the furnace is shown in the preceding table:

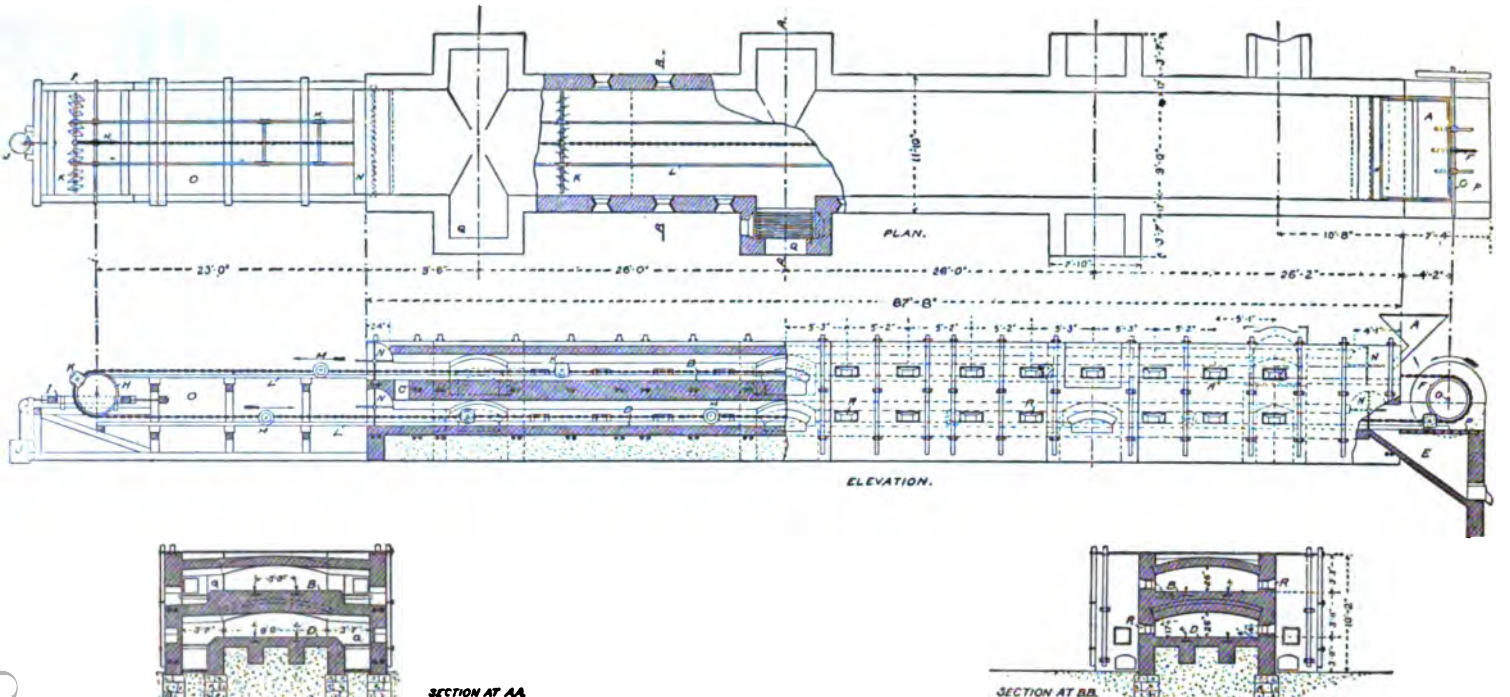
The ore remains in the furnace from 12 to 14 hours. From 1 to 2% of fluedust is ordinarily made. The following partial analyses of fluedust collecting near the furnace and the stack from roasting mixed sulphides are of interest:

Fluedust collecting near—	SiO_2.	FeO.	Cu.	Pb.	Ag, oz.	Au, oz.
Furnace	5.0	39.0	3.0	2.0	37.10	0.10
Stack	.....	.....	.....	4.0	49.30	0.20

Another mechanical furnace of Mr. H. F. Brown which has been roasting white metal satisfactorily in England for several months is a straight-line single-hearth furnace in which the channel-beams supporting the roof, as in the Horseshoe furnace, rest on cast-iron columns held in place by buckstays and tie-rods. The spaces between the columns are closed by sheet-iron doors lined with asbestos mill-board to prevent radiation. The sheet-iron is fitted with angle irons to keep it straight and the doors are held firmly in place by levers forced down between the buckstays. By this arrangement the entire sides of a furnace can be laid bare if necessary. The stirring mechanism consists of carriages which extend entirely across the hearth and are driven by two chains or cables actuated by sprocket wheels. The return track of the carriages is placed over the furnace.

§ 62. The O'Hara Furnace.—This furnace is a long straight-line two-hearth mechanical reverberatory roaster heated from exterior fireplaces. The ore fed automatically at the flue-end of the upper hearth is transferred by means of rabbles to a slot at

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FIGS. 134 TO 137.—THE ALLEN-O'HARA ROASTING FURNACE, PLAN, ELEVATION AND CROSS-SECTIONS.

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the opposite end, through which it drops on to the lower hearth to be carried in the opposite direction to the discharge, located near the feed. Upper and lower hearths have doors at the ends. The rabbles are attached to triangular frames which are pulled over the hearths by an endless chain passing at the ends of the furnace over pulleys, one of which is connected with the driving power. By placing the pulleys some distance from the furnace, chain and rabbles are cooled after leaving it by traveling through the air. While the furnace does excellent work as far as the elimination of sulphur is concerned, the great disadvantage is that the chain and rabbles being dragged over the hearth are quickly worn out, and the rabbles cutting into the floor tear it up, especially if they meet any obstructions. Owing to this the original O'Hara furnace had to be given up. The improvements of Allen and of Brown having removed the difficulties, the furnace took a new lease of life.

The Allen-O'Hara Furnace, represented in Figs. 134 to 137,* has two hearths *B* and *D*, each 90 ft. long by 9 ft. wide. It can be heated from five pairs of fireplaces *Q*, two on the upper and three on the lower hearth; only three, however, are used at one time. The grates being 5 ft. long by 2 ft. 4 in. wide, give a ratio of hearth to possible grate area of 23 : 1. The wear of plows, chain, and hearth has been overcome by laying a pair of rails *L'*, 3 ft. 5 in. apart, on each hearth. These carry six trucks *K* supporting the rabble arms, each with 14 rabbles of steel plate to which white-iron shoes are riveted. The weldless-link chain connecting the trucks is driven by the sprocket-wheel *F* and kept taut by weight *J* acting upon the sliding box *I* of sheave *H*. In order to prevent its sagging and to permit exchanging it in smaller sections, it is attached to trucks *M*, placed midways between the plows. The seven rabbles of one-half of a plow are set in a direction opposite to those of the other half, so as to prevent the truck from being forced off the track. With the next plow the positions will be reversed. Extra rabbles prevent any ore from accumulating on the rails. As there are no rabbles in the center of the hearth because they would be in the way of the chain passing over the sprocket-wheel *F* and the sheave *H*, the ore would form a ridge there and wear out the chain passing over it, if this accumulation were not prevented by

* Peters, "Modern Copper Smelting," 1895, p. 202.

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swinging double-bladed scrapers attached to the backs of the carriages. Two cast-iron plates bolted together at the ends, with sleeves on the bolts to keep them apart, have each near one end a circular opening for a pin, near the other a vertical slot to be used as a guide. The blades of the scraper are guided by arms reaching into these slots, and the handle swings on a pin passing through the circular openings. Thus traveling over the upper hearth, one-half of the scraper removes the central ridge of ore, and when it reaches the tightening sheave, it is raised out of the way; on the lower, it accomplishes the same work with the other half and is again lifted out of the way, when the chain passes over the sprocket-wheel. The ends of the hearths are closed by horizontal turnstile doors *N*. The wheels *F* and *H* being placed at a distance of 23 ft., the chain and plows have a chance to cool. The tie-rods passing through the floors are protected by 2-in. pipes. Beneath the sprocket wheel *F*, is a grating *P*, on which chain and plows can be changed. The ore fed automatically at *A*, receives in its passage over the upper and lower hearth the air necessary for oxidization through dampers in the side doors *R'*, and is discharged into the hopper *E*. The materials required for the erection of the furnace are: Cast iron, 36,000 lb.; wrought iron, 30,000 lb.; firebrick, 8,000; red brick, 125,000; the cost of the furnace at Butte, Mont., is $6,000.

The furnace has so far not been used on mixed sulphides or on lead ores. The ore roasted at Butte* is a mixture of pyrite, chalcopyrite, chalcocite, and bornite, with some enargite, and shows the following average composition: SiO_2, 23%; Cu, 14.6%; Fe, 23%; Zn, 2%; S, 37.4%; Ag, 6.5 oz.; Au, trace. It ranges in size from $1\frac{3}{4}$-in. down to slimes. The coarse ore decrepitates upon heating. The charge lies about 3 in. deep near the feed and remains two hours in the furnace, which puts through from 50 to 60 tons of ore in 24 hours or 67 tons per square foot of hearth area. The roasted ore retains 11.4% of sulphur, and the amount of fluedust formed ranges from $\frac{1}{2}$ to 1%. The carriages make the circuit in $3\frac{3}{4}$ minutes, thus stirring the ore every 37 seconds. Two horse power are required, 100 lb. of coal are consumed per ton of ore, and one fireman attends to two furnaces on a shift. As to the life of the wearing parts, the rabbles with white-iron shoes last one year, the shoes six weeks, the

* Private communication, by C. M. Allen, January, 1897.

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flange-wheels have to be replaced in about six months, the round bushings for the wheels in five weeks, the hexagonal bushings in 10 weeks, and the chain with drop-forge links once a year.

The Brown-O'Hara Furnace.—This furnace has two hearths 90 ft. long and usually 8 ft. wide. The wear of plows, chain, and hearth has been overcome by Brown by placing the stirrer carriages with chain in a small passage on either side of the hearth, separated from it by a slotted wall as in the Horseshoe furnace. From the carriages rabble arms with their steel-plate rabbles reach halfway across the hearth, and thus convey the ore fed automatically on the upper hearth to the discharge on the lower hearth. As far as the writer is aware only one of these furnaces is in use to-day and this at the works of the Pueblo Smelting and Refining Co., Pueblo, Colo. The hearth is 90 ft. long and 11 ft. 3 in. wide; the furnace is heated with oil. The ore when leaving the lower hearth is conveyed by the rabbles to a hopper placed over a circular sinter hearth 12 ft. in diameter. The furnace charge is made up of pyrite and galena, or matte and galena, so as to contain about 10% silica and 20% lead. On leaving the roasting hearth the sulphur has been reduced to 6%, and when drawn from the sinter hearth to 3%. The ore passes in from 8 to 10 hours through the roasting furnace, which puts through 35 tons in 24 hours. The fuel consumption is large.

The following table gives some general facts about the work that has been done with the furnace:

THE BROWN-O'HARA FURNACE.

Length of single hearth (feet)	60	80	90
Width of single hearth (feet)	8	8	8
Hearth area (square feet)	960	1,280	1,440
No. of fireplaces in upper hearth	1	1	2
No. of fireplaces in lower hearth	2	3	3
Length of single grate (feet)	5	5	5
Width of single grate (feet)	3	3	3
Grate area (square feet)	45	60	75
Ratio of hearth to grate area	21⅓ : 1	21⅓ : 1	19⅛ : 1
No. of carriages	4	4	6
No. of blades on a carriage	16	16	16
Carriage makes the circuit in (minutes)	2.66	3.2	3.4
Ore stirred every (seconds)	40	32	34
Horse-power required	2.5	2.5	3.5
No. men on 12-hour shift	1	1	1
Tons of coal per ton of ore	0.1	0.1	0.1
Character of ore—mixed sulphides.			
Approximate composition of ore, SiO_2 25, Fe 15, Pb 8–15, Cu 2, Zn 8, S 25–40.			
Screen-size of ore	4-mesh.		
Depth of charge near fluebridge (inches)	4½ to 6		
Ore remains in furnace (hours)	12		
Ore roasted in 24 hours (tons)	15 to 18	20 to 25	25 to 30
Ore roasted per square foot of hearth area (pounds)	31 to 37	31 to 39	34 to 40
Sulphur in roasted ore (per cent.)	2.7 to 4.5		

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In view of the high percentage of lead in the charge at Pueblo it is difficult to prevent its caking and adhering to the hearth bottom. After many trials it was found cheapest to pay no special attention to this, but to allow the ore to build at the same rate that the rabbles wore off, and to shut down the furnace and cut out the crust when the rabbles had been completely worn down, which occurs twice a year.

§ 63. THE KELLER AUTOMATIC ROASTER.—The latest form of this furnace,* as represented at the Germania Lead Works, Salt Lake City, Utah, is shown in Figs. 138 to 141. It is essentially

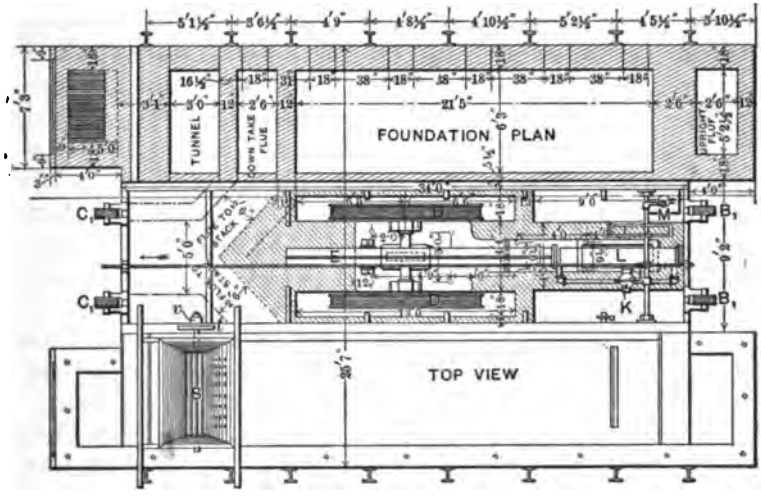


FIG. 138.—THE KELLER AUTOMATIC ROASTER.

the same as the one used in Butte, Mont., under the name of Keller-Cole-Gaylord furnace, patented in 1891, differing from it only in some details of construction. The roaster consists of two rectangular shaft-like furnaces, 8 ft. apart, worked as a pair (Fig. 138). They occupy a floor space of 35 ft. by 42 ft. 3 in., and are, including the feed hoppers, 29 ft. high. Each furnace has five superimposed hearths with slots, these being alternately at one end and the other, so that the ore can be transferred by mechanical means from the feed at the top-hearth to the discharge at the lowest one. The roof of the top hearth

* U. S Patent No. 579,711, March 30, 1897.

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serves as a dryer. Each furnace has one fireplace for heating the ore on the top-hearth quickly to the temperature at which it begins to roast, the heat set free by the oxidation of the sulphides being sufficient for the process to go on without the further use of carbonaceous fuel. The sulphurous gases, traveling in the furnace in the opposite direction from the ore, meet on the top hearth the products of combustion from the fireplace and pass off with them through a flue inclosing the end of the furnace opposite the fireplace and the bottom, and thus reduce the loss of heat by radiation to a minimum. The ore of the pair of furnaces is moved by machinery placed between them. The arrangement of hearths, the manner of firing, and the road the ore travels are shown in Fig. 139. The hearths have the general arrangement in common use with the shelf-burners for roasting pyrite smalls in the manufacture of sulphuric acid. A hearth is 21 ft. 5 in. long, 6 ft. 3 in. wide, 20 in. to the spring of the arch, and this 8 in. high in the center. The hearths are red-brick arches covered with silicious tailings which form the working bottom. The fireplace is located on the ground floor (see also elevation, Fig. 140) and the flame rising through a flue at one end is turned near the top into the roasting hearth and ignites the ore on the top shelf. At the same time it dries and warms the ore spread out on the roof previous to its being put into the furnace through the trapdoor *a*. From the upper hearth the products of combustion from the fireplace travel with the sulphurous gases through flues *b* and *c* and an underground main to the stack. The roasted ore is discharged at E (see also cross section, Fig. 140). As the moving of the ore requires that there shall be a continuous slot passing through the inner walls of a pair of furnaces, the manner of supporting them, and through them the upper hearths, becomes of vital importance. It is shown in Fig. 140. The inner walls are built against 24-in. I-beams *f* running the whole length of the furnace and forming the skewbacks of the arches of the hearths, which in their turn support the floors. At the ends they are riveted by means of angle-irons to 12-in. I-beams *g* laid in the end walls (Fig. 139); they receive additional support from the cast-iron columns *h* (Fig. 140). The I-beams *f* are prevented from bulging out by 6-in. I-beams *i*, serving as distance-pieces. The slots between the I-beams *f*, are closed by small hinged doors *j* (see also Fig.

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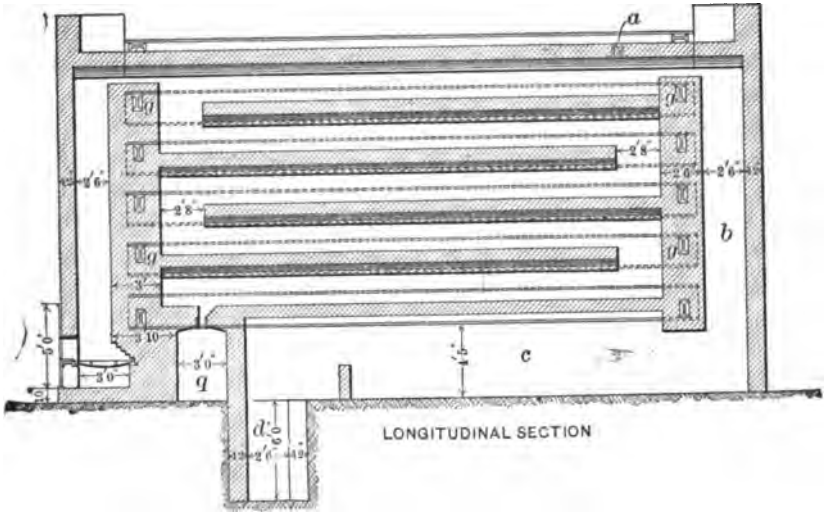


FIG. 139.—THE KELLER AUTOMATIC ROASTER.

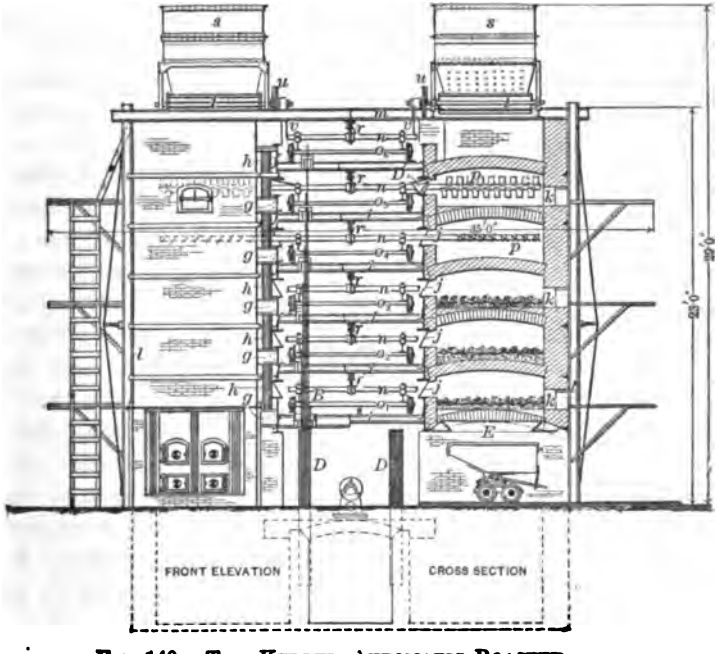


FIG. 140.—THE KELLER AUTOMATIC ROASTER.

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141). These are raised by door-plows D' attached to the stirrer arms and drop again into position, closing the slots when the plows have passed. The outer side walls of the furnaces have each 10 doors, two to a hearth (of which three, k, are shown), to permit access to the hearths from platforms. The walls are well braced by 8-in. I-beams l with hog-yokes; six of them are tied by the usual iron rods, two by I-beams m, as these are to support the feed-hoppers. The movement of the ore (Fig. 140) is effected by six rabble arms n fixed to carriages o_1—o_6 running on rails which are fastened to the I-beams i. The ordinary pipe-arm, which is hollow and has perforations so that air can pass through it into the furnace, has lately been replaced by a telescoping pipe permitting the circulation of water for cooling. The arms passing through the slots in the inner walls are provided with two sets of rabbles p, placed at 180° to each other and mismatched. On reaching the ends of the furnace they are turned 90° by a tripping mechanism, leaving the rabbles in a horizontal position for the return trip, after which they are again revolved another 90° in the same direction, when the second set of rabbles comes in contact with the ore and passing through it, changes the ridges first made into furrows. The quarter-revolution of an arm is caused by a four-point star-wheel q (Fig. 141), in the center, striking at each end of a trip a tumbler r bolted to the bottom of the I-beam i. The ore is fed mechanically on to the roof of the top hearth by the upper stirring carriage. The ore-hopper S is closed at the bottom by a roller-feed t (Figs. 140 and 141), the shaft of which has at one end a ratchet-wheel u with pawl and bell-crank v; the carriage striking the crank turns the roller which feeds a regulated amount of ore, and the crank drops back to an adjustable stop w. The ore is moved by the rake to the opposite end of the furnace until it reaches the trapdoor a (Fig. 139) opening on to the upper hearth. The door is raised by the carriage striking the bent lever x (Fig. 141), to which is attached a weight (not shown), closing it again as the carriage recedes. The motion of the carriage is obtained as follows: Each carriage O_1—O_6 is attached to the ends of a cable, A_1—A_6, passing over two sheaves, B_1—B_6 and C_1—C_6, at the two ends of the furnace (sheaves B_1 and C_1 being six-grooved serve as idle-sheaves for cables A_2—A_6), then around the driving six-groove sheave D with pinion actuated

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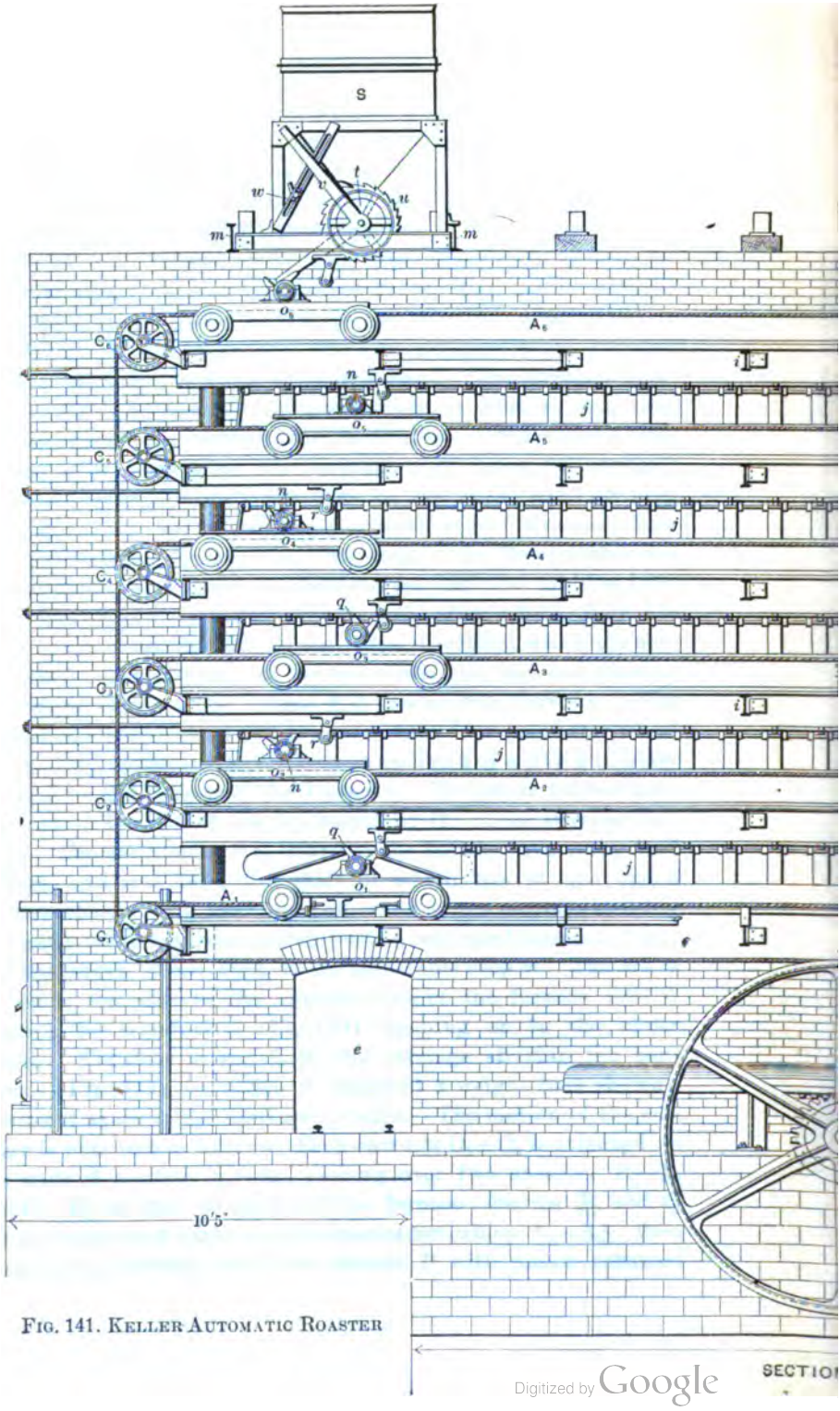
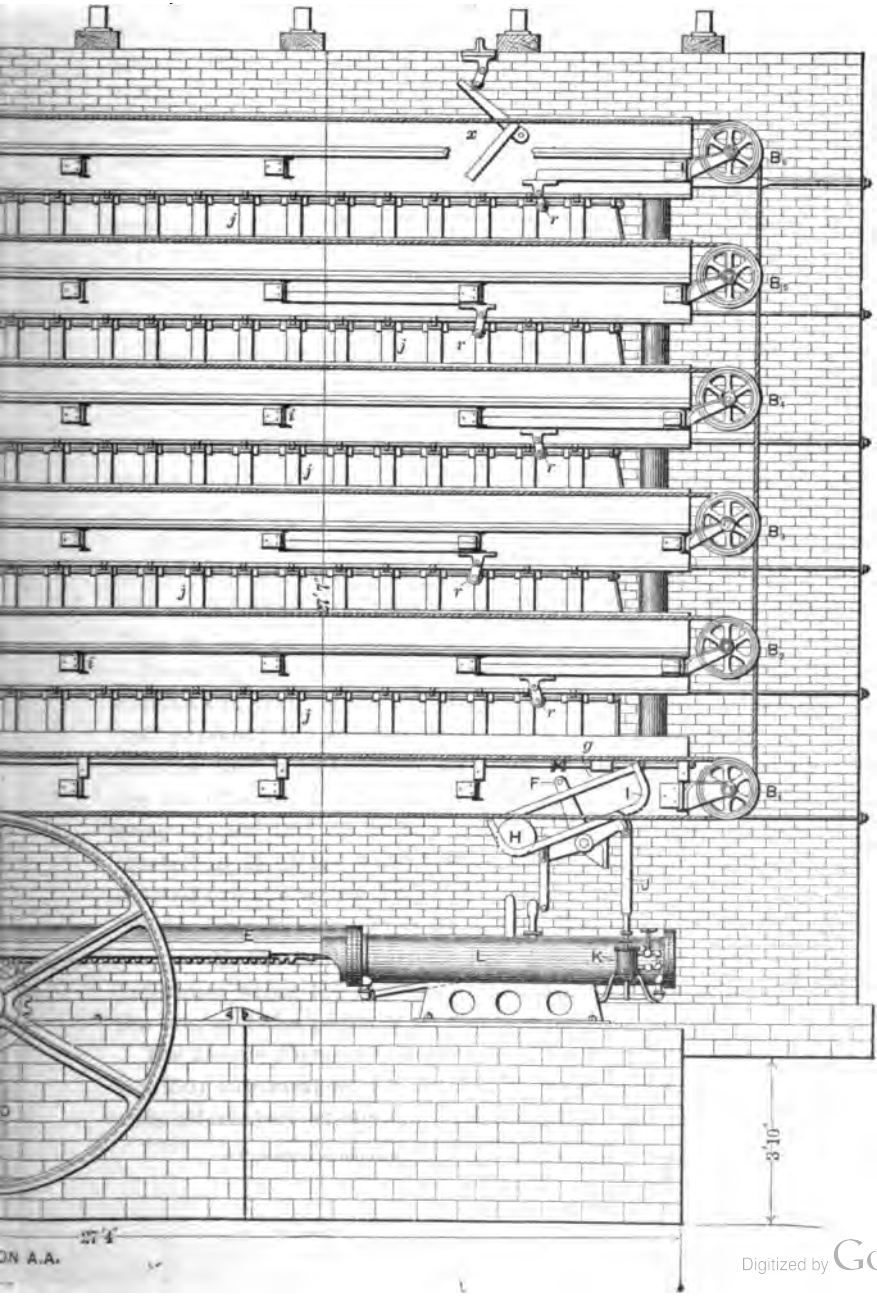


FIG. 141. KELLER AUTOMATIC ROASTER

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SECTION



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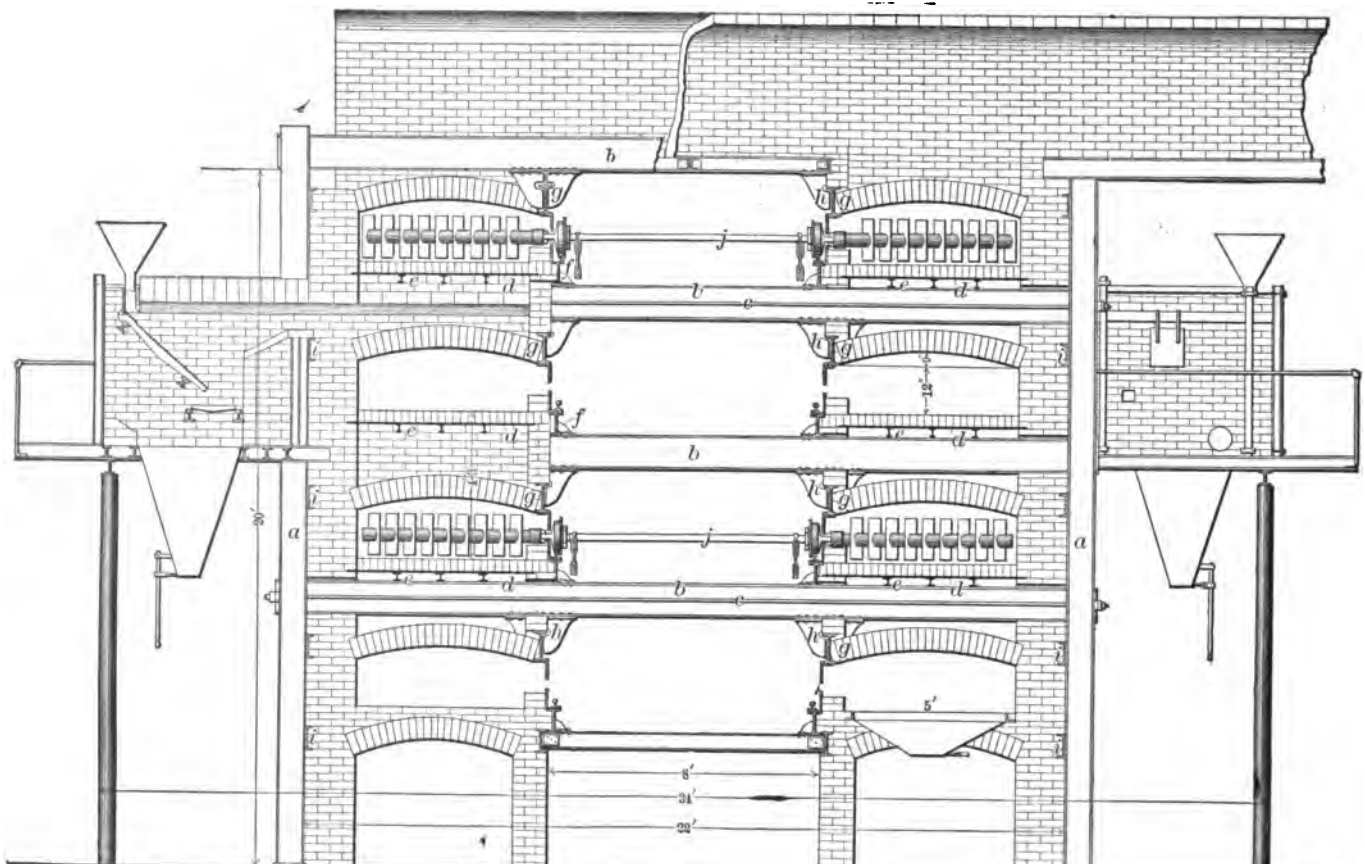


FIG. 142.—THE WETHEY FURNACE, CROSS-SECTION SHOWING FIRE BOX, FLUE, DISCHARGE, CARRIAGE, AND PLOWS.

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top hearth, when moved mechanically to the opposite end, may drop on to the next hearth, then travel in the opposite direction and so on until it reaches the discharge on the lowest hearth. The furnace is built in an iron framework. This consists of vertical channel-beams *a* joined by horizontal castings (not shown) riveted to them, which form the supports for the horizontal I-beams *b* carrying the upper hearths. The channel beams *a* are braced by tie-rods *c*. The brick bottoms of the upper hearths are built on steel plates *d* which rest on small 3-in. I-beams *e*. Along the inner sides of the furnace between each pair of channel-beams *b* run longitudinally two 6-in. I-beams; the lower ones *f* resting on the channel-beams carry the tracks and angle-irons against which the inner walls are built; the upper ones *g* being suspended from the channel-beams by means of the castings *h* form the inner skewbacks of the roof, the outer ones *i* being held in place by the vertical channel-beams *a*. The upper hearths are suspended independently one from another and firmly tied, so that there is no danger of arches or floors falling in.

The ore is stirred and moved forward by the rabble arms *j* attached to endless chains running outside of the furnace and driven by sprocket-wheels similarly to those shown in Figs. 134 to 137 with the Allen-O'Hara furnace, the upper hearths and the lower hearths of the furnace each having an endless chain, so that there are four for the double furnace. The 2-in. slot through which the rabble arms pass is closed by tripping doors. The figure gives a fire-box on the second hearth, the products of combustion traveling in an opposite direction to the ore. Experience has shown that it is better to have the fireplace on the upper hearth near the point where the ore enters the furnace and to have the gases travel with the ore. If it is important, as with lead-bearing ores, to eliminate as much as possible the sulphur, a second fire-box is placed on the lower hearth, the gases traveling in opposite direction to the ore. Thus the gases from the upper fireplace pass off at the side of the third hearth, those from the lower fireplace at the second hearth.

The total weight of the iron work is 170,000 lb.; 122,500 red brick and 2,000 firebrick are necessary; 2 horse power is required to run the furnace, and it costs at Butte, Mont., $10,000.

It has been running at the Butte Reduction Works since 1894. The following are the leading data furnished by Mr

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Wethey: The hearths are 50 ft. long and 5 ft. wide, giving a hearth area of 2,000 sq. ft.; there are four fireplaces with grates 38 by 33 in., giving a grate area in round figures of 80 sq. ft., and a ratio of hearth to grate area of 25:1. There are 8 rabble arms to a furnace, each with 14 rabbles, four of which make the circuit of two floors in 185 seconds, thus stirring the ore once in 46 seconds. The ore treated is a mixture of pyrite, chalcopyrite, and blende, ranging in size from 12-mesh to 4-mesh. Its average composition is SiO_2 10%, Fe 35%, Cu 10%, Zn 5%, S 40%, and Ag (Au) 10 to 12 oz. per ton. The depth of the charge near the feed is about 4 in., the ore remains from 8 to 10 hours in the furnace, and the roasted ore retains about 8% sulphur. The furnace puts through 45 tons of raw ore in 24 hours, consumes per ton of ore 110 lb. of slack coal and 0.1 gal. black oil for lubricating, one man is required per shift, and the fluedust amounts to 0.5% of the ore charged. Another form of the Wothey furnace is:

The Single Furnace shown in horizontal section in Fig. 143. Its general construction is the same as that of the double furnace, the plan representing one of the four superimposed hearths. On each side of the hearth is a slot through which the rabble arm *a* passes, being supported at both ends by carriages *b* running on rails *c*. By this contrivance it has been possible to make the hearth double the usual width, viz., 10 ft., as in the figure, and even 12 ft. This of course greatly simplifies and cheapens the construction. The total weight of the iron work of the furnace is 143,000 lb.; 95,000 red brick and 2,000 firebrick are necessary; 2 horse power is required to run the furnace.

§ 65. The Bruckner Furnace.—The furnace in common use to-day, although differing very much in detail from the one originally introduced by Brückner in 1867* in Colorado, retains the same general characteristics, viz., a brick-lined horizontal cylinder of boiler iron which revolves on friction rollers between a fire-box and a flue and stirs the ore-charge automatically. Figs. 144 and 145 represent a modern furnace. They show a cylinder closed at the ends by truncated cones (heads) revolving with two friction rings on two pairs of carrying rollers. One of the carrying-roller shafts is rotated by a spur-wheel gear driven by a pinion shaft, with a pulley at its end. The fire-box, which

* "Transactions of American Institute of Mining Engineers," ii., p. 295; *Berg- und Hüttenmännische Zeitung*, 1869, p. 130.

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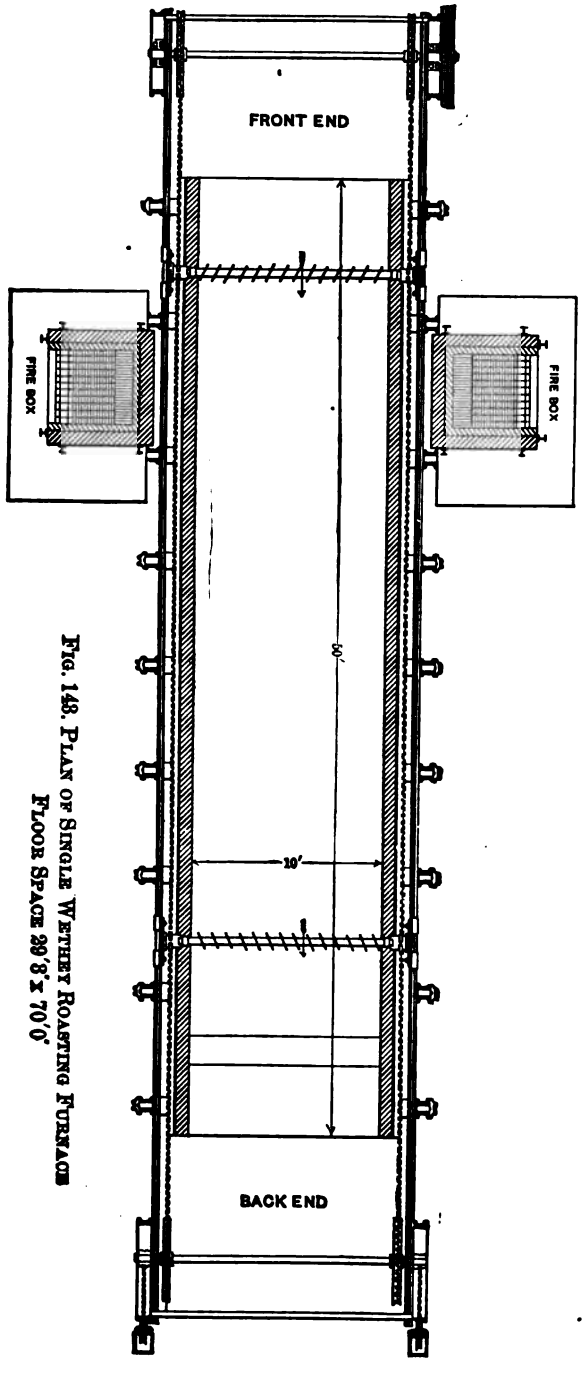
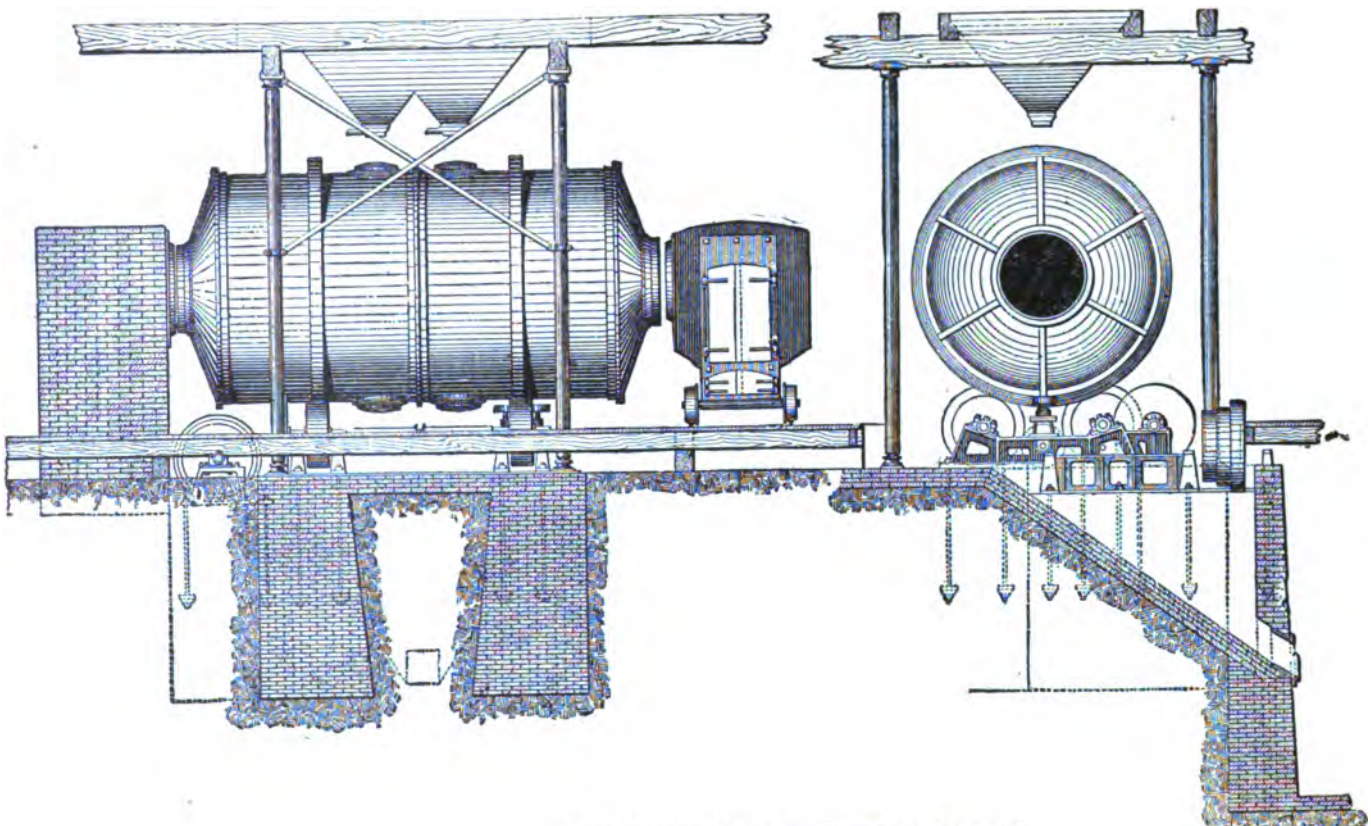


FIG. 149. PLAN OF SINGLE WETHEY ROASTING FURNACE
FLOOR SPACE 29'8" x 70'0"

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FIGS. 144 AND 145.—THE BRUCKNER FURNACE.

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was formerly stationary, has been made movable. It is placed on wheels and can be moved at right angles to the axis of the furnace (as in the figure). Often, however, the track runs parallel with it. With a movable fire-box, air is readily admitted to the roasting charge and all work in the furnace made relatively easy. With the tracks running at right angles to the axis of the furnace, one fire-box can be used for several furnaces. If they run parallel to the axis each furnace must have its own fire-box and this arrangement is common with mixed ores low in sulphur.

A common size of cylinder made to-day is 18 ft. 6 in. long and 8 ft. 6 in. diameter, the largest probably 22 ft. long and 8 ft. 6 in. diameter. Such a cylinder has four discharge openings, two of which are used for receiving the charge dropped from the feed-hoppers. A convenient way of operating the gate of a hopper is by means of a lever and connecting rod worked from a platform on the side of the roaster building, the attendant thus not being exposed to the heat of the furnace. The cylinder is commonly lined with red brick and the heads with firebrick. In treating lead ores or matte, the body of the hearth is often lined with firebrick, if not too expensive; as in barring off adhering parts of the charge red brick is liable to be quickly injured. A furnace 18 ft. 6 in. long and 8 ft. 6 in. in diameter weighs 45,000 lb., requires 100 firebrick for the heads, 2,800 firebrick for the movable fire-box, and 4,300 red brick for the body of the furnace. It costs about $4,000 and requires from 2 to 3 horse power.

The main points in running the furnace, are to adapt the number of revolutions of the cylinder to the character of the ore and to maintain the correct temperature. While in chloridizing roasting of dry silver ores the revolutions may be many, say 15 per hour, with lead-bearing sulphides they have in some instances been reduced to 4 revolutions per hour and even to 1½ because of the slowness with which a particle of lead-bearing material roasts when it rolls down the inclined surface of the charge and the danger of turning it under before the sulphur has burned off.

The manner of firing varies, of course, with the different stages of the process. When a charge of cold and often wet ore has been dropped from the feed-hoppers, the furnace becomes cool and the fire has to be urged to heat the furnace up again and to bring the ore to the temperature at which the sulphides begin to oxidize. The heat thus generated makes it, as a rule, neces-

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sary to stop firing altogether, and when the sulphur has been burned off, extraneous heat has nearly always to be supplied to decompose sulphate and assist in the oxidation of the remaining sulphide. The length of time that an ore will roast without the use of the fire-box depends not only upon the character of the ore and its percentage of sulphur, but on the elevation at which the operation is carried on. At, say, 1,000 ft. above the level of the sea, a pyrite concentrate will require about 8 hours to become sufficiently heated to start the oxidation, when it may roast 36 hours more without any additional firing. At an elevation of say 5,000 ft. the charge will roast without the use of the fire-box, only about 12 hours. In a plant located 10,000 ft. above the level of the sea, where on account of the rarified air the oxidation proceeds very slowly, the fire-box may have to be used three times during the 60 hours it takes to reduce of a charge to 4 or 5%.

The admission of air necessary for oxidation to the furnace has been a source of much trouble. If the furnace is heated with oil atomized by steam, or better by compressed air, there is no difficulty, as, if the opening in the fire-box opposite the cylinder through which the burner is introduced, is made large, enough air will be drawn in. When the furnace is fired with coal it has been found necessary in some cases to introduce a steam or air-jet between the necks of the fire-box and the cylinder and stir up the sea of heavy sulphurous gas floating on the ore that it may be carried off by the products of combustion. Aaron,* in 1882, suggested making the neck of the fire-box muffle-shaped, leaving that of the cylinder circular; thus cold air, heavier than the hot sulphur dioxide, would enter the furnace beneath the flame, and raising the sulphur dioxide, come in contact with the surface of the roasting charge.

One great drawback of the Brückner furnace is the large percentage of fluedust. While this has been diminished by the slower movement of the cylinder, 5% is not an uncommon figure. Assays of samples of fluedust taken near the throat of the furnace and near the stack gave the following figures:

Sample taken near	Cu.	Pb.	Ag, oz.	Au, oz.
Furnace	3.8%	4.5%	89.7%	0.30%
Stack	2.7	2.5	93.7	0.30

* "Eighth Annual Report of the State Mineralogist of California," p. 849.

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The subjoined table gives some of the leading facts about the work done by the furnace:

THE BRUCKNER FURNACE.

Length of cylinder	22′	22′	18′	20′
Diameter of cylinder	8′ 6″	7′	7′	6′
Diameter of neck	2′	2′	2′	2′
Length of grate	4′	4′	3′	5′
Width of grate	2′	2′	2′	2′ 6″
No. of revolutions per hour while roasting	2–3	2–3	2–3	1½
No. of revolutions per hour while discharging	15	15	15	1½
Roasting time of (hours)	48	36	36	48
Using fire-box (hours)	36	24	24	24
Using fire-box (how often)	Twice.	Twice.	Twice.	Once.
Not using fire-box (hours)	12	12	12	24
Not using fire-box (how often)	Once.	Once.	Once.	Once.
Horse-power required	2–3	2–3	2–3	3
The ore: Mixed sulphide with some matte. (*a*)				
Size of ore	4	4	4	4
Tons in 24 hours	9	7	6	8½
Per cent. fluedust	.....	.....	.....	3
Roasted ore: Pulverulent and sintered.				
Per cent. sulphur in	3–7	3–7	3–7	3–4
No. of men in 12-hour shift	½	½	½	2
Pounds of coal per ton of ore	360	360	360	240
Gallons lubricating oil per ton of ore	.....	.....	.....	0.10

MIXED SULPHIDE ORE. (*a*)

SiO_2.	Fe.	Zn.	CaO.	Pb.	Cu.	S.	Ag, oz.	Au, oz.
5.0	28.7	6.0	1.3	14.8	.....	32.2	17.2	.....
.....	.....	.....	.....	22.6	.....	33.2	9.3	.....
2.2	9.3	15.5	0.3	43.5	.....	26.0	29.4	.....
6.0	26.0	.....	.....	18.0	.....	.....	9.1	.....

PYRITE WITH MATTE. (*a*)

8–16	40–30	0–12	.....	0–20	0–4	40–20	6–50	0.3–0

§ 66. SELECTION OF FURNACE SITE AND GENERAL ARRANGEMENT OF PLANT.*—In selecting a furnace site, economical as well as technical considerations must come into play. Some of these are:

(*a*) Are the conditions such as to justify the erection of a permanent plant, or is the structure to be only a temporary one? If the former, the plant is built of stone, brick, and iron; all arrangements are so planned as to allow enlargement by simply adding to the old plant. If the plant be a temporary one, as little money as possible will be spent on the building, which is usually a light wooden one. Often the necessary capital is not available to build properly from the start, and a temporary structure has to grow by its own profits into a large, permanent plant.

* Locke, "Smelting Plants," Cincinnati, 1883; Austin, "Transactions of American Institute of Mining Engineers," xxvi., p. 388; Vezin, *Ibid.*, p. 1095; *Engineering and Mining Journal*, July 31, 1897.

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In such a case the general plan will commonly be the same as that of the permanent plant, the temporary structure being replaced as opportunity permits, although in some instances it may prove more advantageous to erect the permanent plant independently of the temporary one, this continuing to run till the other is ready to be started up.

(*b*) What is the precise nature of the work to be done? Do the ores arrive in large or small quantities, at regular or irregular intervals, and through the whole year or only during a part of it? Upon these points it will depend whether the ore, as it comes, can be directly conveyed to ore-beds, whether bins are sufficient to store it, or whether a storage yard has to be provided. If the last, the ore may generally lie uncovered, but sometimes it has to be protected from snow. For fuel, protection is always needed. The size of the plant and the richness of the ore will determine whether hand sampling, mechanical sampling, or a combination of the two, is most desirable. The character of the ore must decide whether it will be necessary to provide a place for ore-roasters and their crushing plant.

(*c*) Should a sloping or level site be chosen? Until quite recently the answer would have been unhesitatingly a sloping site, but the balance seems now to incline in favor of a level site. The disadvantages of a terraced site are: a great cost of plant on account of necessary excavations and retaining walls; great extension of plant, as the location of its different parts has to be regulated by the slope; want of accessibility, involving much tramming and handling; difficulty of supervision; and lack of ventilation. Advocates of a level or nearly level site claim: cheapness of plant, as there is little or no excavating and building of retaining walls; compactness of plant, as the different parts can be placed side by side and at elevations convenient for use; free accessibility with little tramming and hardly any handling, as the materials are collected in elevated bins; ease of supervision; abundance of ventilation.

The cost of elevating is small (*e.g.*, at Denver ½c. per ton to 30 ft.). In a terraced site several small elevators are often necessary to handle the intermediary products; in a level site the same number of large elevators built to serve different levels may do all the work. Well-constructed elevators do not get out of order more easily than other indispensable machinery and should be

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duplicated, as the blowers are. In a modern iron blast furnace plant, handling very much larger amounts of materials than a lead or copper smelting plant, the level site has entirely displaced the terraced site.

(*d*) What water supply can be had for boiler and jackets? If there is no natural flow from an elevated point, a well will have to be sunk or the water obtained from a creek below, which requires pumps and additional boiler power to raise the water to the tank. The character of the water, if hard or soft, clear or muddy, may also be an important consideration. The general arrangement of a plant will depend mainly on whether the site is inclined or level.

If there is a good natural fall, the arrangement will be as follows: taking the furnace floor as base line, there will be on one side the slag dump, with a fall of 30 or more feet; on the other, the furnace, reaching to the feed floor, 18 or more feet above. The roasting furnaces will be on the third level, if the ore is all to be roasted; otherwise at the height of the feed floor. Next comes the track from which the crushed ore is discharged into the hoppers of the roasters, eight or more feet above the roaster floor and below the discharge of the sampling mill, through which most of the ores that come to the smelter pass. It will be the level of the ore yard and the storage place for fluxes and fuels, the latter, however, being more frequently assembled on or slightly above the level of the feed floor of the blast furnace, the coke some distance away on account of the small amount required and as a precaution against fire.

The precise way in which the floors are placed will vary with the configuration of the ground, as it is necessary not only to have the right fall, but also convenient grades for bringing in the materials and carrying away the products.

The situation of the machinery for driving the blowers, elevators, and dynamos, and that of the pumps and of the machine and blacksmith shops, also varies, although it is usually on the furnace floor. The machinery for crushing is all found in the sampling department. The steam is usually all furnished from one set of boilers. The office and laboratory are ordinarily near the ore yard.

If there is no fall, the cars bringing ore, flux, and fuel will come into the works elevated on trestle-work that the materials

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may be readily unloaded into bins and thence discharged into trucks. Most of the tramming will be done on the ground and the elevators be so located as to be central for the work they are destined to do; the same will be the case with the machinery collected in *one* building and put under the charge of a head mechanic.

With all smelting plants the machinery ought to be duplicated, as delays caused by repairs or break-downs are very costly, the same general expense of management going on whether a plant is working to its full capacity or not.

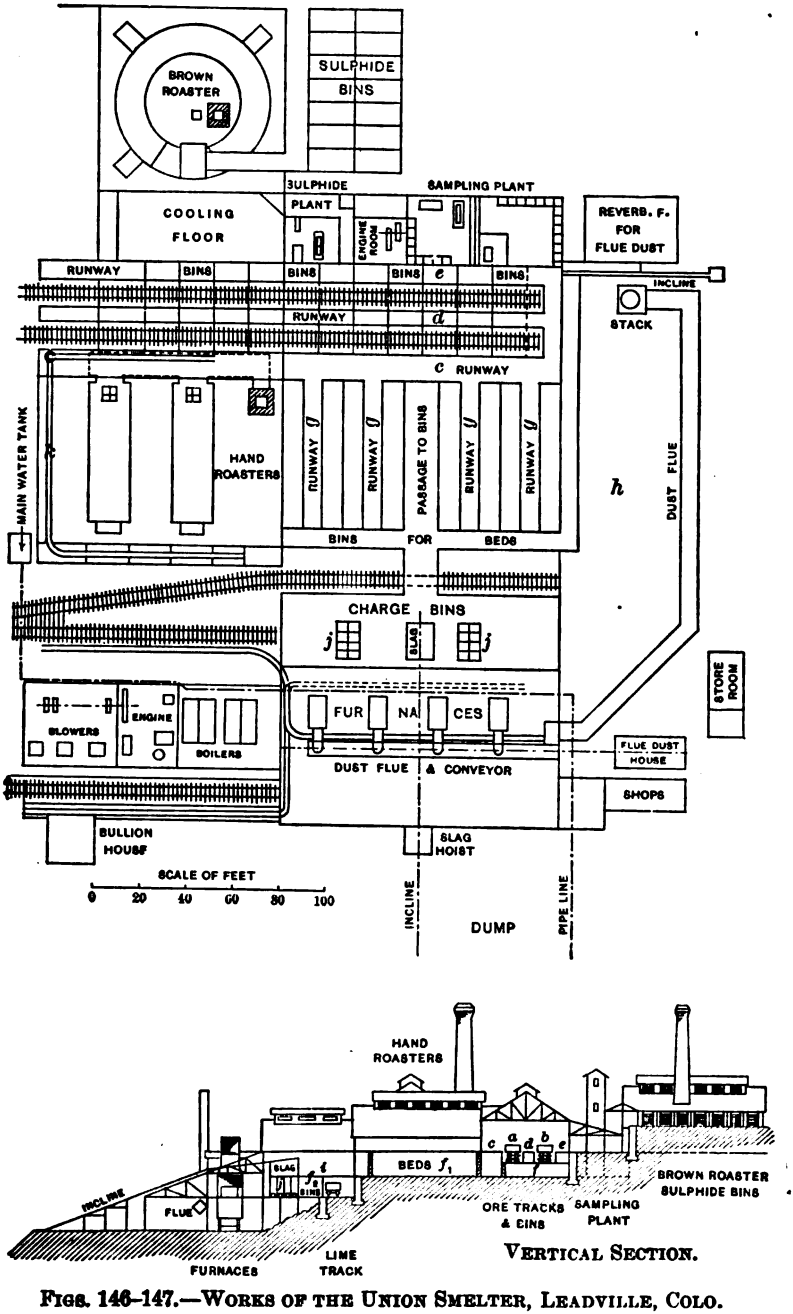
A few examples of plants in successful operation may serve to illustrate the general points discussed:

The Works of the Union Smelter, Leadville, Colo.—This plant, situated on the south side of California Gulch, is shown in plan and vertical longitudinal section in Figs. 146 and 147.* It is built on a very steep hillside, having a slope to the north. The vertical distance from the blast-furnace floor to the floor of the Brown Horseshoe furnace is 70 ft. for the short horizontal distance of 318 ft. from the exterior side of the blast-furnace building to that of the Brown roaster building; the width is only 124 ft. The plant is very compact, requiring little tramming of materials.

The rails *a* and *b*, on which the cars bringing the ores run, are laid on trestle-work. They are placed sufficiently low for the bottom of a car to be on the same level with the three runways *c*, *d*, and *e*, along the sides of the tracks. The ore is sampled by fractional selection. The rejected portion that is to be smelted raw is discharged into bedding bins *f*, under the tracks, or wheeled over runways *g* to make up the beds *f'*. If the rejected ore is to be roasted, it is crushed and rolled in the sulphide plant and elevated to the sulphide bins, whence it is run over an elevated track to the feed-hoppers of the Brown roaster or of the hand roasters, the latter being on the same level as the ore-bedding bins. The cooling floor of the Brown furnace is on the level of the sampling floor, that of the hand reverberatory furnaces on the lime track. The first sample obtained by fractional selection is reduced down to the assay sample in the sampling department.

* The drawings were made specially for this book by Mr. F. E. Shepard, of the Denver Engineering Works.

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Figs. 146–147.—Works of the Union Smelter, Leadville, Colo.

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Fluxes and coke arrive on the lime track; limestone is unloaded into bins *f''* on the feed floor; iron ore is piled up on the paved platform *h* on the east side; bituminous coal for the boilers comes in on the bullion track; that for the hand roasters on track *a*. It is unloaded into trucks running on the track *k* suspended from the roof trusses of the hand-roaster building and discharged into the bins below. The coal for the Brown roaster is wheeled to it from track *b*, a short distance; coke is unloaded from the railroad cars and discharged on the feed floor; what has to be stored is heaped up under cover on the east side.

From the beds the ore mixture goes in two-wheeled barrows over the runways *i* to be dumped into the charge bins *j*, of which there are four to a furnace. Between the two sets of charge bins is the slag bin, which receives its material from the dump by an inclined hoist. From the necessary materials assembled on the feed floor the charges are made up for the blast furnaces. The base bullion produced is loaded on the lower track; the matte is broken by hand, loaded on gondola cars on the bullion track, shifted by a locomotive to track *b*, and discharged in front of the crushing and sampling plant. The slag goes to the dump and the gases pass through a sheet-iron flue suspended in front of the furnaces into a brick flue and then into the stack. The dust that collects in the iron flue is discharged every two days by means of a conveyer into the fluedust-house, whence it is wheeled to the incline to be hoisted to the feed-hopper of the reverberatory furnace and melted down, the gases passing off into the blast-furnace stack.

On the east side of the blast-furnace building and on the level of the furnace floor are the boilers, engines, and blowers; on the west side are the shops and storeroom, as well as the office and laboratory, the latter two not being shown in the plan.

Coming to details, the sampling and sulphide plants are separated by the engine-room. A slide-valve engine, which receives steam from the boiler plant, furnishes the necessary power.

The floor of the sampling plant is covered with iron plates. The plant consists of a Blake crusher, 7 by 10 in., a pair of Cornish rolls, 14 by 27 in., a sample grinder and a sample dryer. This is a wrought-iron steam jacket, 4 by 10 ft., the steam being maintained at 25 lb. pressure by means of a Curtis reducing valve, the main steam line carrying 85 lb. There is no

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elevator in this plant, the samples being raised to the mouth of the crusher on an incline in a large self-dumping car, thus saving much time in cleaning the plant between samples.

The sulphide plant has a 9 by 15-in. Dodge crusher, discharging into a rubber-belt elevator with steel buckets which deliver the ore into a trommel, 3 ft. in diameter and 6 ft. long, with a 2-mesh screen. The oversize passes into the feed-hopper of a pair of 16 by 36-in. finishing rolls. The sulphide department is small, because most of the ores treated require no roasting. The 14 sulphide bins are 10 by 20 ft. and 8 ft. high. The Brown Horseshoe furnace is of standard size (see § 61). Its engine receives steam from the main boiler plant. The two hand reverberatory roasting furnaces have a hearth 60 by 14 ft.

The blast-furnace department has 4 water-jacket blast furnaces 33 by 100 in. at the tuyeres and 14 ft. high from tuyeres to feed floor. Blast is supplied by three No. 6½ Baker blowers and one No. 7 Cycloidal blower.

Steam for the entire plant is supplied by four 48 in. by 14 ft. horizontal tubular boilers. The main engine is an 18 by 42-in. single-cylinder, non-condensing Hamilton-Corliss engine, coupled direct to the main blower-shaft. The engine serving to hoist the slag has an 8 by 10-in. steam cylinder, the car is operated by a double-engine friction hoist with cylindrical drums and wire rope.

Water is supplied from a ditch emptying into the main water tank. There is also a duplicate plant of duplex direct-acting steam pumps in California Gulch, drawing from a crib placed below the level of the creek. The pumps are connected with the fire system.

Light is furnished by a 25 K. W. General Electric generator driven by a 35 horse power Westinghouse compound engine.

*Works of the United Smelting and Refining Co., East Helena, Mont.**—The general plan and section of the works are represented in Figs. 148 and 149, the detail arrangement of the blast-furnace building and the slag-hoist in Figs. 150 and 151. The works are built on a gently sloping hillside; their main extension, 607 ft., is at right angles to the slope, which is of 4° 40′, there being a fall of 21 ft. in 258 ft., the horizontal distance from upper receiving track (No. 2) to outer edge of blast-furnace

* Drawings and data furnished by Mr. A. L. Dean, superintendent, September, 1896.

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room. The large number of roasting furnaces indicate that the works are planned to treat principally sulphide ores, although considerable amounts of oxide ore are worked. Sulphide ores arrive on the upper track (No. 2), pass through a special mechanical sampling department into sulphide ore bins, whence they are discharged when they are required for the reverberatory roasting furnaces or the covered roasting stalls. The roasted ore is collected in cooling bins, whence it goes to the blast furnaces. The oxide ores arrive on tracks Nos. 4 and 5 and are sampled by fractional selection; the rejected ore is collected in the oxide bins, while the sample goes to the oxide department of the sampling mill to be cut down either mechanically or by hand as may be necessary. There are a number of storage bins for iron ore along the main track near the axis line. The fuel for boilers, roasters, and blast furnaces arrives on the elevated tracks Nos. 4 and 5, beneath which are bunkers to receive it. Ore, flux, and coke are elevated to the feed floor of the blast-furnace room to be charged into the blast furnaces. The bullion produced is shipped from track No. 6; the waste slag is poured over a pouring plate into Nesmith cars and drawn to the waste-slag dump on the depressed slag-track; rich slags are crushed and raised by an inclined slag-conveyer and discharged into a slag-bin, from which they are taken to be resmelted either in the ore or the matte blast furnaces. The matte is loaded on flat cars or coal cars and hauled by a switch engine to the sampling mill. The general arrangement shows the plant to be very compact and requiring but little tramming and handling of ores and products.

In the sampling house the power is obtained by a 14 by 36-in. Fraser & Chalmers Corliss engine, the boxes of the main shaft resting on brick piers in the basement of the building. Steam is furnished from the one boiler plant. There are two departments, one for sulphide, the other for oxide ores. The sulphide department has one 9 by 15-in. Blake crusher, one pair 36-in. Chicago Iron Works belted rolls, one pair 26-in. Fraser & Chalmers geared rolls, two large elevators, two trommels with $\frac{3}{4}$-in. and $\frac{1}{4}$-in. screens, and two Taylor and Brunton samplers. Floors, chutes, hoppers, and spouts are all lined with $\frac{1}{4}$-in. or $\frac{1}{2}$-in. steel plates. All sulphide ores are mechanically sampled. From the ore that is to go to the roasting stalls everything under $\frac{3}{4}$-in. is screened out. In the oxide department hand sampling

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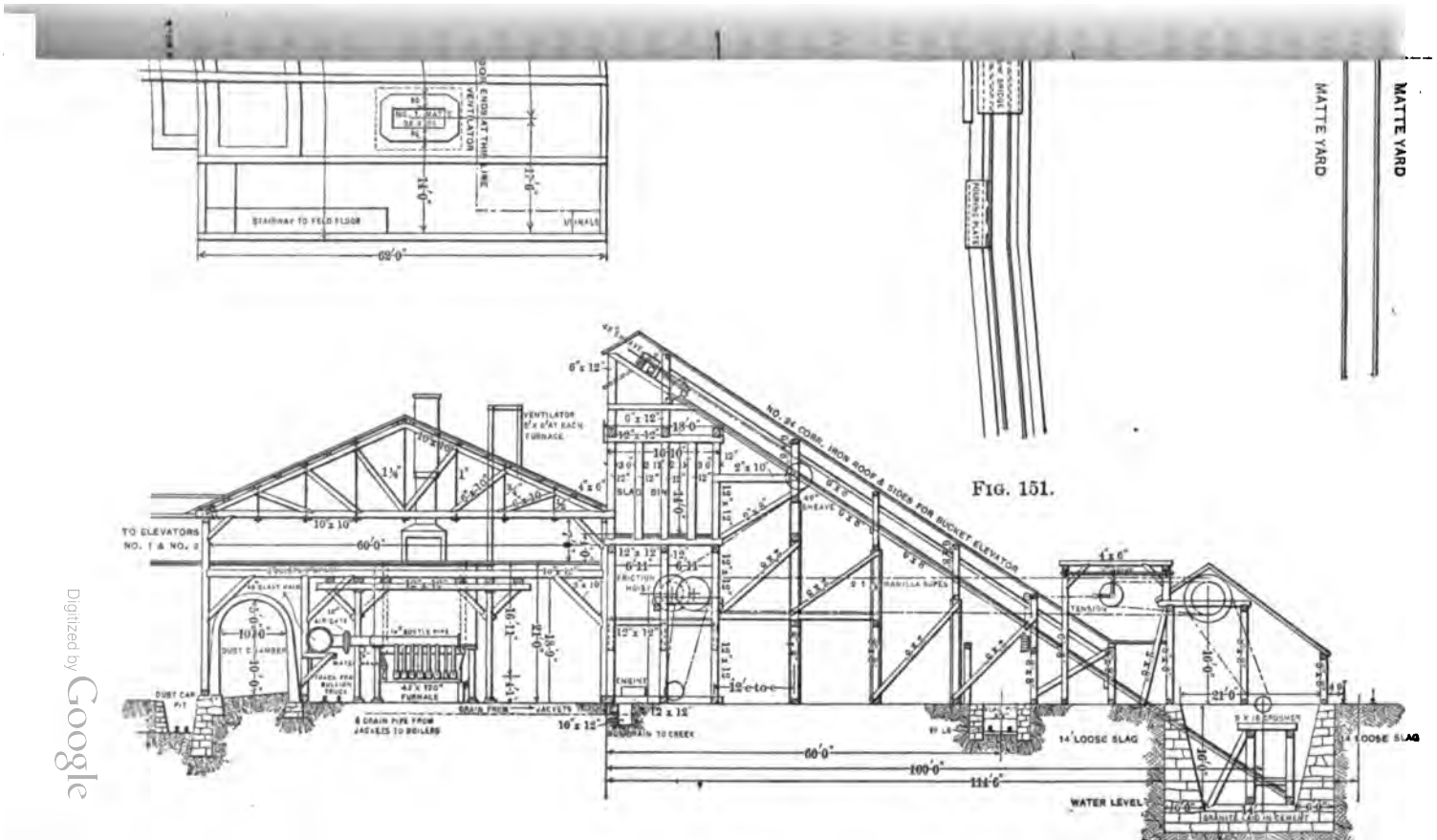


FIG. 151.

FIGS. 148 TO 151.—WORKS OF THE UNITED SMELTING AND REFINING CO., EAST HELENA, MONT.

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as well as mechanical sampling are practiced. The machinery for hand sampling consists of one 7 by 10-in. and one 4 by 10-in. Blake crusher and one pair of 12 by 12-in. rolls; that for mechanical sampling of one 9 by 15-in. Blake crusher with elevator, and a Taylor and Brunton sampler; finally, there is a sample grinder. The floor is covered with ¼-in. steel plate. In this department all the limestone required by the works is crushed and the matte crushed, rolled, and sampled.

Roasting is carried on in four different places. The south calciner building has eight long-hearth hand-reverberatory roasters without fuse-boxes, having hearths 48 by 15 ft.; the north calciner building has eight similar furnaces with 52 by 15-ft. hearths, lack of room in either case forbidding greater length. The Brückner room has two cylinders 18 ft. 6 in. by 8 ft. 6 in., and lastly there is a block of 20 covered roasting stalls, 18 of which are 15 by 10 ft. and two 20 by 10 ft.; they are all 5 ft. high.

The blast-furnace department (Figs. 150 and 151) contains four ore furnaces and one matte furnace, each of which is provided with a hood and ventilating fan to take away the fumes from the front when tapping slag and matte. The charges are weighed on five sets of Howe multiple-beam charging scales and are raised to the feed floor by two Frisbie elevators driven by wire rope. The blower-room has five No. 6 and one No. 5 Root blowers, one No. 3½ Baker blower, and one small Sturtevant fan for the blacksmith forges. When the plant had only four blast furnaces and a smaller number of blowers each blower served one furnace; to-day they all blow into one 48-in. main serving the five furnaces. Power on the dump was furnished by a 9 by 12-in. slide-valve engine to the 9 by 15-in. Blake crusher and the skip which receives the discharge from the crusher and raises it to the top of the incline. To-day it is received from the main engine through the line shaft running blowers, elevators, etc.

The water supply comes from a reservoir about 1,200 ft. from the works, the dam of which is a cribbing filled in with blast-furnace slag. From the reservoir the water is conducted through a box-flume of 2-in. tamarack lumber, 20 by 12 in. in the clear, laid from 2 to 4 ft. underground. It is provided with a screen at the throat and has a water-gate and two settling tanks. The first tank, which is 20 ft. long, has riffles and can be uncovered

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and readily cleaned out. The fall from the mouth of the flume to the second settling tank is 14 in. in 1,000 ft. when the water is lowest in the creek. This tank is open. There are arrangements for running into it sufficient hot water from the overflow of the jackets and condenser to keep it from freezing. From the tank the water runs into a well (25 ft. in diameter) over which is placed a No. G Knowles fire pump. The pump is about 65 ft. distant from the buildings and is protected by the 15-in. retaining wall of the well. Steam connection is made directly with the boilers by means of a 3-in. pipe. The pump is fitted with Locke's pump governor, by means of which sufficient steam is admitted to keep the water at the pump and in the 6-in. fire-main at a constant pressure of 40 lb. In case of an emergency the steam supply can be increased to 200 lb. A second fire pump, Knowles No. E, is located in the basement of the engine-room. It is used to supply cold water to the water jackets of the blast furnaces and the condenser of the two 22 by 48-in. Corliss engines. It may be cut off from the two blast furnaces and condenser and used as a second fire pump. The fire-main is a 6-in. cast-iron pipe, the fire hydrants are for 2½-in. hose fitted with 1-in. nozzles.

Power is furnished by a single battery of six Fraser & Chalmers tubular boilers, each 16 ft. by 54 in., with 144 3½-in. tubes; the pressure carried ranges from 75 to 85 lb. The water is supplied by one No. 5 Knowles boiler feed pump and one No. 2 Körting injector; it can, however, be secured directly from the fire-main, should this be necessary. Usually the overflow from the water jackets of the blast furnaces, with an average temperature of 52° C., furnishes the necessary supply. The water flows through a filter, 8 by 4 by 3 ft., filled with pieces of clean coke, and is then pumped into a 42-in. by 14-ft. live-steam purifier, whence it passes into the boilers at a temperature of about 120° C. Each boiler, in addition to the usual accessories, is provided with a Hotchkiss purifier. In the engine-room there is one 22 by 48-in. Allis-Corliss and one 22 by 48-in. Fraser & Chalmers Corliss engine; both are connected with a condenser by Knowles' automatic relief valves. They are run alternately and drive the blowers, elevators, dynamos, Brückner cylinders, slag crusher and hoist, machine shop and ventilating fans.

The electric light plant contains one 250 incandescent lamp U.

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S. dynamo, one 10-arc light U. S. dynamo, one 40-arc light Standard Electric Co. dynamo, lamp regulators, lightning arresters, etc. One 7 by 12-in. automatic cut-off engine is used in case of shut-downs. Finally, the machine shop contains the following tools: One Lodge-Davis swing lathe, one drill-press, one pipe-cutting and threading machine, one Wiley & Russell bolt-cutting and threading machine, one Doty power punch and shear, one emery wheel, grindstone, Trimmer's forming machine, etc. Most of the cars are built and nearly all the sheet-iron work is done at the works.

Works of the Montana Smelting Co., Great Falls, Mont.—These works, planned by Mr. A. Eilers, are shown in plan and vertical section in Figs. 152 and 153. They are built, like the plant at East Helena, on a gently sloping hillside, the angle of slope being 2° 20′, or 46.7 ft. in 1,050 ft. from the upper floor of the sampling works to the tapping floor of the blast furnace, to which must be added the height of the dump, which is over 30 ft. The longest extension here is on the line of slope, while with the previously mentioned plant it is at right angles to it. The works are planned to treat considerable amounts of sulphide ore, and as during the winter months there is liable to be a lack of ore special attention has been given to the storing of large quantities of it, sulphide ores being collected above the roaster building, carbonate ores below it. The loaded cars, arriving on the main track, are weighed on track-scales and then switched off on side tracks, which lead to the "crushing house and sampling works," to the "bins for sulphide ores" (tracks 9 and 10), and to the "bins for carbonate ores" (tracks 2 to 6). At the sampling works all ores requiring crushing are received, and the sulphide bins are filled with concentrates, which are sampled by fractional selection while they are being unloaded. The lower bins are for ores, sampled in the same way, that go straight to the blast furnace. Track No. 1 brings the coal for the boilers and takes the base bullion and matte produced in the blast furnaces. Foul slag is raised to the feed floor by an elevator.

The crushing house and sampling mill, which is fitted up with the necessary machinery for crushing, sampling, and grinding, delivers the sulphide ores over a tramway to their respective bins.

In the roaster building is room for 20 calcining furnaces (see § 58, Figs. 96 to 101) and fusing furnaces. They deliver their gases into two parallel flues (Fig. 101) running along the

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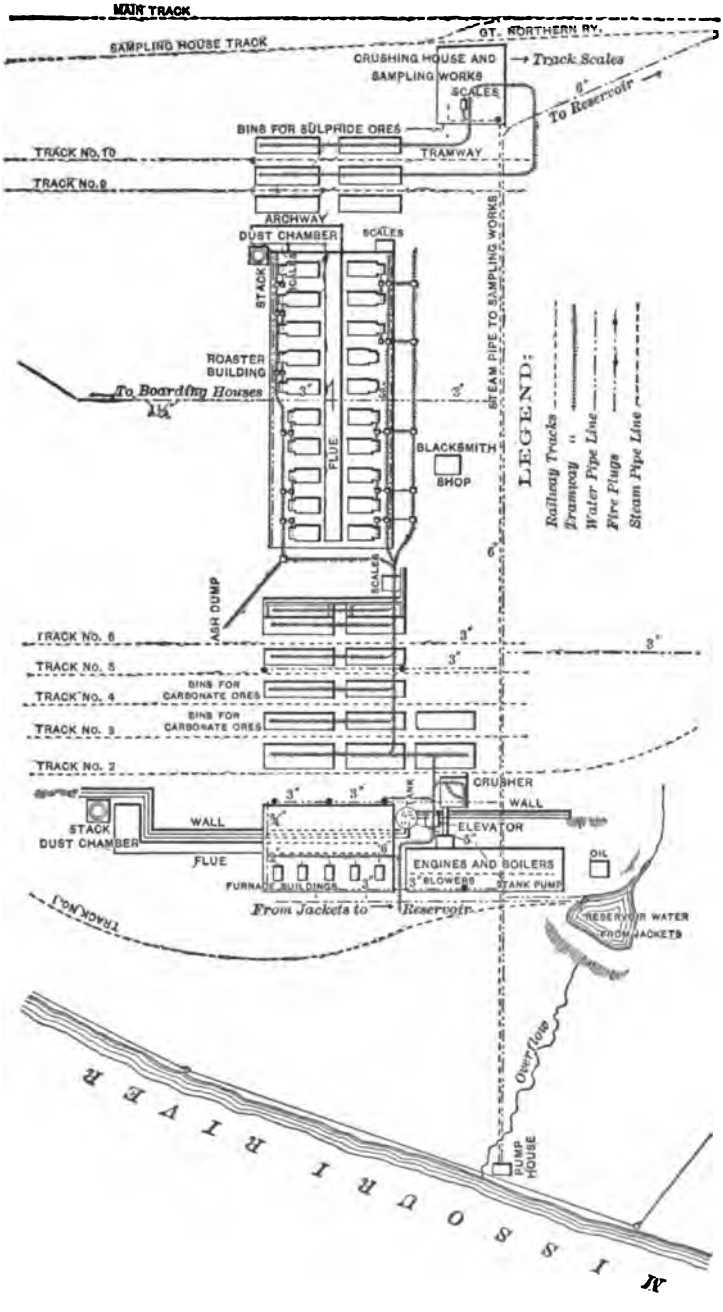


FIG. 152.—WORKS OF THE MONTANA SMELTING CO., GREAT FALLS, MONT.—PLAN.

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center of the building (Fig. 152). These are built one against the other and are combined into one main flue after they have received the gases of the single furnaces. This passes out of the building and terminates in a series of dust chambers, connected with the stack by a small flue. The tramways are for carrying ore, fuel, and ashes. The floor of the roaster building is on the same level as the top of the lower ore bins; the roasted ore can thus be easily discharged into them.

The blast-furnace building shows three floors instead of two, as is usual. The upper or ore-bedding floor is on the same level as the bottom of the bins for carbonate ores. From the beds made on it the ore mixture is dumped on the feed floor near the single furnaces, where the charges are made up. The fluxes, arriving on track No. 2, are passed through the crusher if necessary; the fuels arrive on the same track. On the furnace

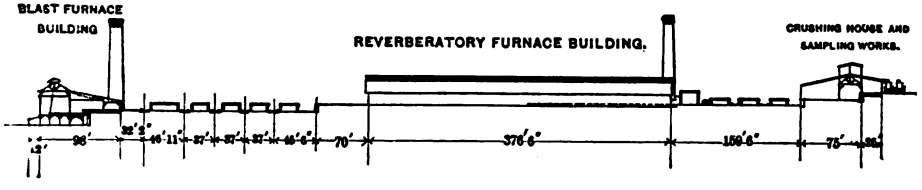


FIG. 153.—WORKS OF THE MONTANA SMELTING CO., GREAT FALLS, MONT.—VERTICAL SECTION.

floor is room for five blast furnaces, the details of which are shown later on. The furnaces face toward the river, where the ground is high and furnishes an ample slag dump. The furnace gases pass through a wide flue, back of the furnaces, leading into a series of dust chambers, which are connected with the stack. The details of flue and chambers are given in § 93. The ventilation of the furnace floor shown in Fig. 153 is the one strongly recommended by Eilers. It consists in allowing the feed floor to extend only to the front of the furnaces and closing it off from the entire front part of the building by a wooden partition, which slants backward to the ridgepole of the roof. Thus a hood as long as the furnace building, and having a width of about 12 ft., reaches from the level of the feed floor to the ventilator and draws off all the vapors and smoke from the tap-hole, the slag-pot, and the lead-well. A similar arrangement at the smelting-works in Clausthal,* Prussia, was very efficacious in

* Private notes, 1890.

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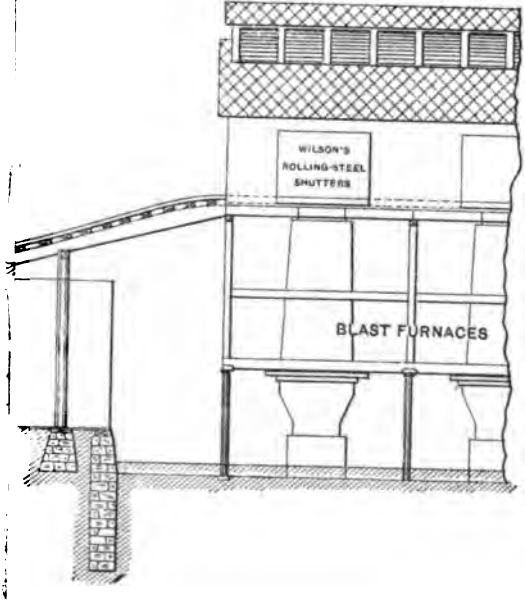
removing the danger of lead poisoning, which had previously been a common thing.

Next to the furnace-room are placed the boilers, engines, blowers, etc. The drawing, Fig. 152, shows the blacksmith shop, the pump-house, the water-pipes and fire-plugs, the water-tank and also the lower reservoir, which receives the overflow of the jackets, but not the upper reservoir.

Works of the Pueblo Smelting and Refining Co., Pueblo, Colo. —Fig. 154 shows a vertical section through the smelting department of these works, drawn approximately to scale. They are seen to be built on a hillside and to have a considerable extension along the slope. The main point of interest is the simple manner of handling the ore and the mechanical feeding of the blast furnaces. The ores are classed under two heads, as ores that can be smelted raw and ores that require a preliminary roasting. The first class arrives on the elevated unloading track for ore and is sampled by fractional selection. The sample is discharged to the right on an elevated platform in the sample mill and held in separate stalls until it is passed either through a crusher or through a chute on to the main floor, where it is cut down by hand, usually by quartering. The rejected ore is unloaded to the left into bins with a sloping bottom, whence it is discharged into Hunt automatc dump cars* and distributed over the ore-bedding bins, on the ground floor of the sampling mill, each holding about 2,000 tons of ore. The bottom of a Hunt car has the form of an inverted V sloping toward the inclined sides which swing open, when released by a lever, and allow the ore to be discharged on either side of the inclined track *b* running over the center of the bin. The car is attached by a cable to a counterpoise which is heavier than the car when empty and lighter than when it is filled with ore. Thus the car will run down the incline when filled and up again to its original position when emptied. The unloading is effected automatically by an adjustable stop along the side of the track which strikes the lever and thus releases the sides. By changing the position of the stop, the car can be emptied at any point along the center line of a bedding-bin, and the ore is then spread out in the usual way by hand. As one man can attend to several cars, the labor involved in bedding the ore is reduced to an extremely low figure.

* *Colorado State School of Mines Scientific Quarterly*, vol. ii., No. 1, p. 35.

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Ores that are to be roasted are crushed and rolled in the sulphide mill and sampled like the raw-smelting ores. The rejected ore is raised by an elevator and stored in sulphide bins *c*, whence it is discharged and run into the feed-hoppers of the Brown-O'Hara mechanical roasting furnaces (see § 62). The slag-roasted ore is transferred into the Hunt car to be spread over the ore beds.

Fluxes and fuels arrive on a separate track. The sample, when taken, is collected in small bins *d* discharging into a car running to the sampling mill, while the rest goes into the flux and fuel bins below.

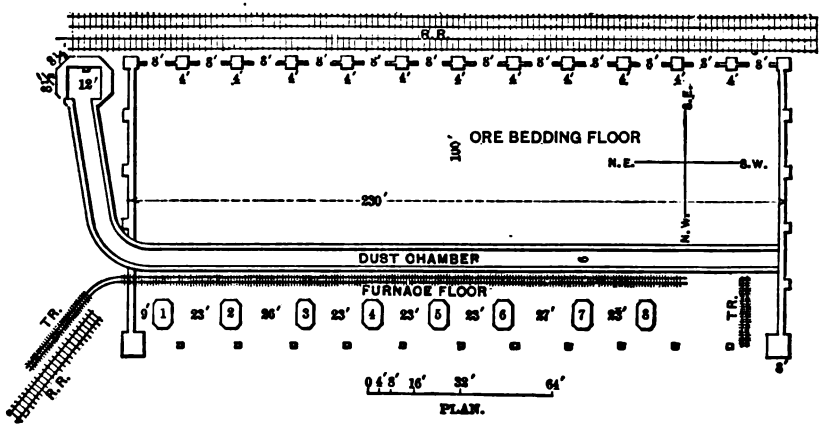
Thus the ore is assembled on one side and flux and fuel on the opposite side of the pit *e* of the single inclined hoist, which raises the charge to the feed floor of a row of seven blast furnaces. The charge, weighing from 4 to 5 tons, is dumped in at the ends of the car, the coke being added last to prevent its being crushed. The car and the charge distributors are those patented by Williams.* The car resembles very much that of Hunt. The distributor castings, having the form of an inverted V, are placed some distance above the level of the charge parallel to the width of the furnace. The car, raised by machinery, is stopped when it arrives at the throat of the furnace, the two doors closing the latter are lowered, and the charge is dropped. In descending it strikes the distributors and is thus evenly divided over the surface.

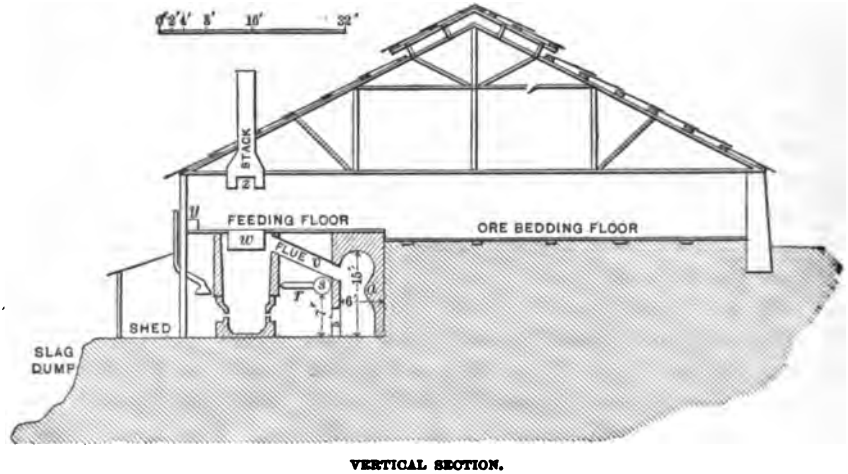
At the tuyere-level the blast furnaces are 60 in. wide between sides or 48 in. between the projecting water-cooled tuyere-nozzles, and 120 in. long. They have a working height of 20 ft. and have each six 4-in. tuyeres on either side. The slag-dump is at right angles to the plane of the illustration. The gases pass through Williams dust collectors (see § 93) and brick flues to the stack.

Works of the Globe Smelting and Refining Co., Denver, Colo. —The blast-furnace department of these works is erected on a steep hillside. It has only two levels: those of the feed floor and the furnace floor. The general arrangement, as planned by the superintendent, Dr. M. W. Iles, is based on the following scheme: if two intersecting lines are taken, running east and west and north and south, the ground level will be represented by

* U. S. Patent No. 554,562, Feb. 11, 1896.

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VERTICAL SECTION.

FIGS. 155–156.—WORKS OF THE OMAHA AND GRANT S. AND R. CO., DENVER, COLO.

Nos. 1-8.—Blast furnaces.
(*a*) Projection in the dust chamber (abolished).
(*r*) Bustle pipe.
(*s*) Induction pipe.
(*v*) Flue carrying fluedust from the blast furnace into the dust chamber.
(*w*) Sheet-iron curtain through which the charges are fed into the blast furnace (abolished).
(*y*) Elevated tramway for the fuel trucks that the contents may be dumped on the feed floor.
(*z*) Telescope stack, used in blowing in or out, to carry explosive or hot gases out into the open air (abolished)
(*RR*) Broad-gauge track on feed floor on which ore and fluxes arrive.
(*TR*) Narrow-gauge track on furnace floor delivering the base bullion to the broad-gauge track.
(*TR'*) Tramway for raising slag, etc. (altered).

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the southeast and southwest fields, containing respectively the furnaces facing south in a row (from which the gases pass into a main flue leading to the bag-house, located toward the east) and the boilers and machinery for blowing, lighting, etc. The upper fields will show in the northeast field the calcining furnaces, and in the northwest field the sampling department. Two sets of tracks on the upper level, running east and west, bring in ore, flux, and fuel on either side of the sampling and calcining departments; another track on the furnace floor takes away the base bullion produced. An inclined elevator running north and south brings the foul slag, matte, and fluedust from the furnace floor to an elevated track between the calcining and sampling departments and dumps the three products in the places where they are to be further treated, *i.e.*, the slag near the ore beds, the matte near the sampling mill to be crushed before roasting, and the fluedust near the fusing furnaces, where it is to be slagged.

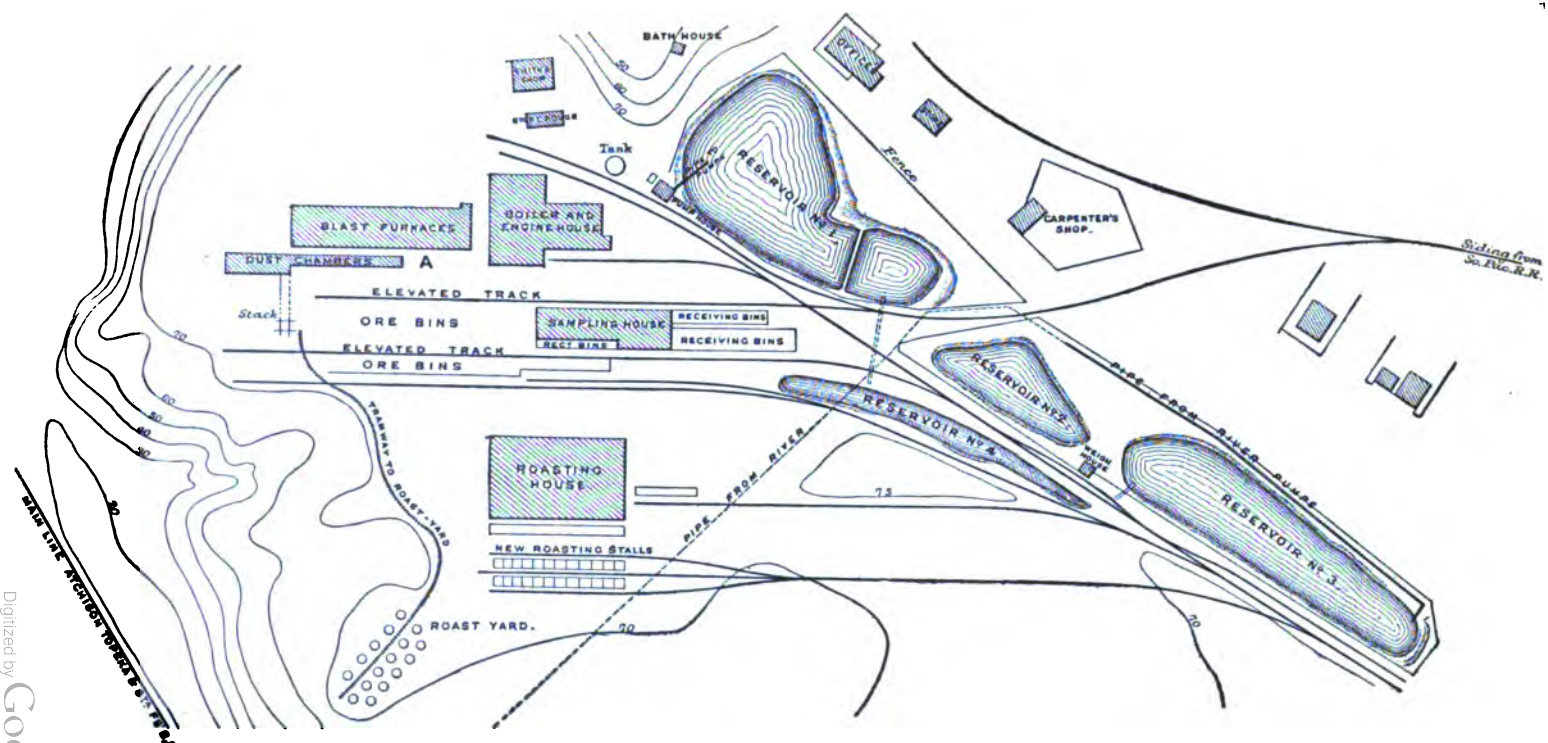
Works of the Omaha and Grant Smelting and Refining Co., Denver, Colo.—The general arrangement of this plant is similar to that at the Globe works. Figs. 155 and 156 represent a plan and a cross-section of the blast-furnace building. There have been numerous changes in detail since the drawings were made, but the general outline remains the same.

To be noted especially is the large ore-bedding floor. It is the practice to make two large ore beds, each occupying nearly one-half of the floor, about 8 ft. high and holding about 3,000 tons of ore. All the furnaces receive their ore from one bed, and while this is being consumed the other bed is made.

Another feature not to be overlooked is the position of the fronts of the furnaces in regard to the points of the compass. Facing northwest they are as much as possible in the shade, an important consideration in hot weather.

The sheet-iron hood z', placed in front of each furnace to carry off the fumes that arise on tapping the slag, never did its work satisfactorily. The hood now ends in a horizontal pipe which either terminates in the dust-chamber or in a galvanized iron pipe, common to a number of furnaces, and connected with a fan which sucks off the fumes and discharges them into the open air. Sheet-iron plates hung on either side of the hood prevent the draft on the furnace floor from carrying off the fumes into the building before they can be taken away by the hood.

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Scale 1/8 in.—100 ft.

FIG. 157.—WORKS OF THE CONSOLIDATED KANSAS CITY SMELTING AND REFINING CO. AT EL PASO, TEXAS.

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*Works of the Consolidated Kansas City Smelting and Refining Co., El Paso, Tex.**—This smelting plant, shown in Fig. 157, is erected on level ground which falls away suddenly on one side, thus furnishing a natural dump.

The ore arrives on an elevated track and is sampled by fractional selection while it is being unloaded into the ore bins or the receiving bins. The ore or bedding bins, holding from 100 to 250 tons of ore, are supplied from regular shippers. The smaller receiving bins, 60 in number, containing from 10 to 20 tons, hold consignments whose values have not yet been settled upon. Most bins are 8 ft. high. As the amount of sulphide ore treated is small, the roasting-house contains only four long-hearth hand-reverberatory roasting furnaces. The blast-furnace building contains eight stacks, 42 by 100 in. and 17 ft. high from tuyeres to feed-floor, placed in a row. They receive their blast from a main, 42 in. in diameter near the blowers and 30 in. at the third furnace. The gases pass through long dust-chambers into the stack. As all the furnaces run on the same charge, this is made up from a centrally located spot where the scales are placed. The runways between the bins have tracks from which branches lead into the bins proper. Near the scales are small charge bins holding the materials from which the charge is made up. They are constantly replenished from the mixture bins, receiving bins, etc., from which the tracks converge toward the charge bins. Two charging wagons, one with ore and flux, the other with slag and fuel, making up a charge, are hoisted together in a double elevator to the feed floor, and go to one of the eight blast furnaces. The base bullion produced is sampled and shipped; the slag goes to the dump, where most of it is broken for railroad ballast; the matte used to be roasted first in heaps in the roast-yard and then in the hand reverberatory furnaces; now the roast-heaps have been replaced by stalls. The boiler room contains five tubular boilers of 80 horse power, run at a pressure of 90 lb.; the engine-room, one Hendy and Meyer 100 horse power, and Porter-Allen 125 horse power engine used alternately. These drive the machinery in the sampling house, the blowers (Baker, two No. 7½, one No. 5½, one No. 4½; Root, one No. 7), etc. There is a special engine for the dynamo, generating electricity to light the works and the offices. Water is

* Collins, *Proceedings of the Institute of Civil Engineers* (England), 1893, cxii., p. 140.

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pumped from the Rio Grande, 63 ft. below the level of the works. The pump-house at the river (not shown) contains two horizontal double-plunger Dean pumps with a 12-in. steam cylinder and 7-in. plunger, and a 14-in. steam cylinder and 10¼-in. plunger respectively, the stroke being 10-in., which gives a total capacity of 750,000 gal. in 24 hours. The water from the blast-furnace jackets runs into four cooling reservoirs, whence another set of Dean pumps in the pond pump-house (12-in. steam cylinder with 7-in. plunger, and 10-in. steam cylinder with 6-in. plunger, 10-in. stroke), with a daily capacity of 650,000 gal., delivers it into a tank 10 ft. above the top of the jackets.

Small Blast-Furnace Plant.—In Figs. 158 and 159 is represented a small, compact smelting plant for treating sulphide and carbonate ores. It has a single blast furnace large enough to justify putting up a mechanical sampler. The drawings are taken from a plant erected in British Columbia by Messrs. Fraser & Chalmers, Chicago, Ill. The works are built on a hillside, and have two main levels, an upper one *a* (Fig. 159), on which the ore arrives in railroad cars, and a lower one *b*, the furnace floor; adjoining this is sufficient fall for the dump. The plan, Fig. 158, shows the general arrangement of the works. Along the upper track are located the sampling mill *c*, with the carbonate ore bins *d* to the right, discharging on the feed floor of the blast furnace, and the sulphide ore mill *e* with the sulphide ore bins *f* to the left. These are close to the Brown Horseshoe roasting furnace and sufficiently elevated to permit the ore to be discharged into a truck and run into the feed-hopper of the roaster. Roasted ore, matte, and foul slag are loaded on railroad cars on the bullion track and brought to the upper or ore track. Back of the furnace is the engine-room with ordinary slide-valve engine and blower and adjoining is the boiler-room. The one engine drives all the machinery of the plant.

Carbonate ore arriving on the upper railroad track is unloaded into the carbonate ore bins *d*, and at the same time sampled by fractional selection. The sample is received in a small car standing on a narrow-gauge track and run to one of the sample ore-bins *o*. From these it is discharged into a small car *P*, on the level of the feed floor, and transferred to the hopper of the 7 by 10-in. Blake crusher *A* to be discharged into the mechanical feeder *B* and then to pass through the geared 24 by 14-in. rolls

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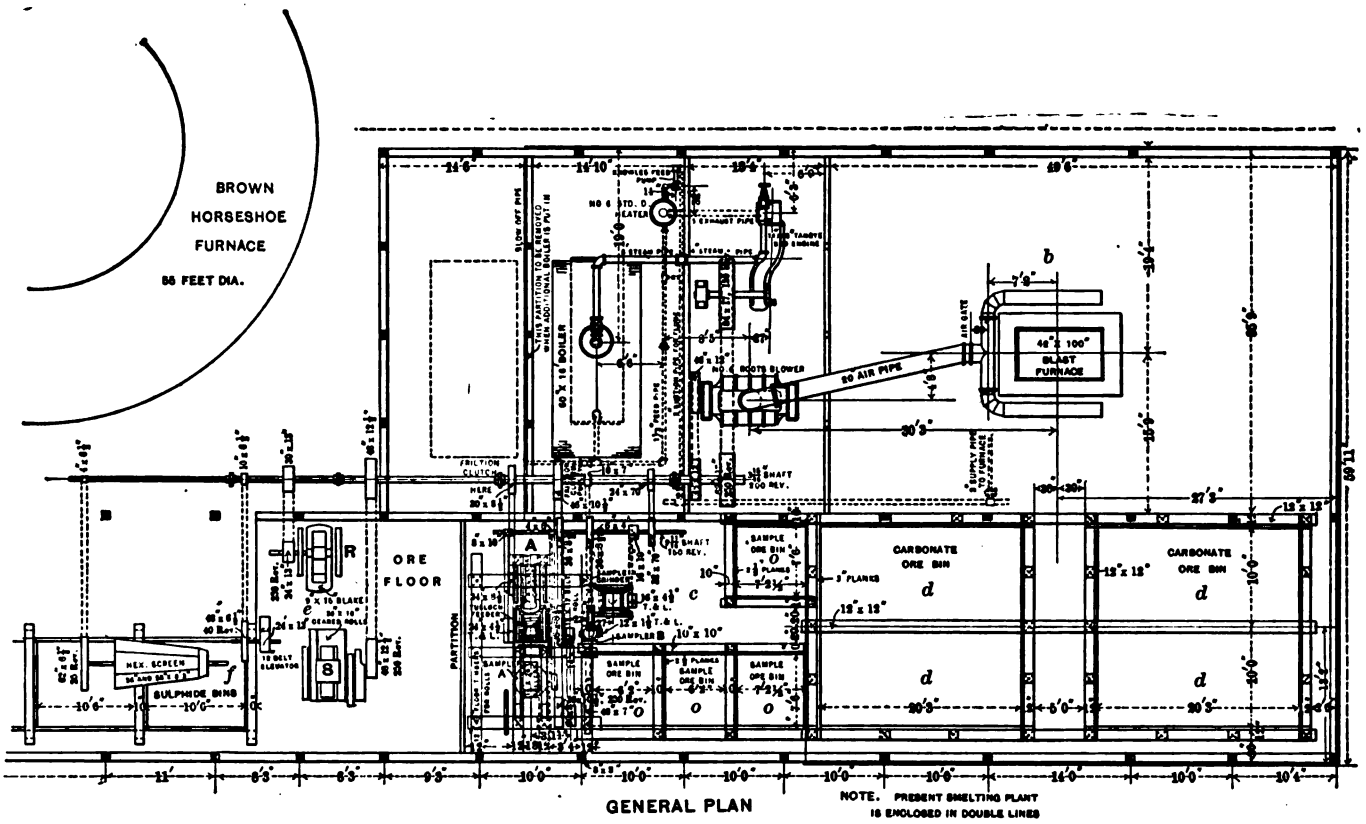
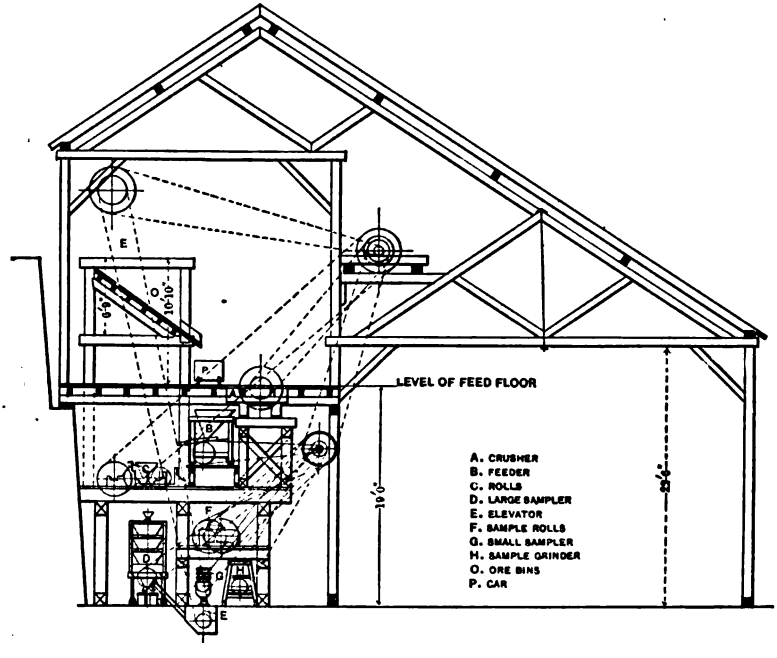


FIG. 158.—SMALL BLAST FURNACE PLANT.

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C into the large Bridgman sampler *D*. The rejected ore from this machine goes into the boot of the elevator *E*, which returns it to the sample bin from which it came. The sample obtained from the large sampler is rolled in the 12 by 12½-in. belted rolls *F* and passed through the small Bridgman sampler *G*, whence the discard passes into the boot of the elevator to be returned to the sample bin, while the reduced sample is ready to be ground in the sample grinder *H*.



ELEVATION OF SAMPLING WORKS.

FIG. 159.—SMALL BLAST FURNACE PLANT.

Sulphide ore arriving on the upper track is passed through the 9 by 15-in. Blake crusher *R*, then through the 30 by 16-in. geared rolls *S*, and the hexagonal screen, the oversize being returned to the rolls.

§ 67. THE BLAST FURNACE AND ITS ACCESSORY APPARATUS.—*Introductory Remarks.*—The blast furnaces used in lead smelting

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are of the most varied description. Taken in cross section they are square, polygonal, circular, oblong, and elliptical. In vertical section they are prismatic or with sides tapering toward the bottom, and in addition may or may not have a bosh. Then, the smelting zone may be inclosed by water jackets and the crucible may be internal, or partly internal and partly external. Furnaces with a detached crucible have been constructed and patented,* but are not, so far as the writer is aware, in use. Finally, the lead may either be tapped from the bottom of the crucible or removed by means of Arents' automatic tap.

Only two kinds of furnaces are now in use in this country. One has an internal crucible, the other a crucible partly internal and partly external; both have water jackets and Arents' automatic tap. They may be either circular or oblong. The circular furnace has the form of an inverted truncated cone, and is used for smelting small quantities of ore or by-products of refining works; the oblong furnace has sides that are either vertical or very slightly inclined; it always has a bosh, and is the common ore-smelting furnace. Oblong furnaces have almost entirely replaced circular furnaces.

In the subjoined legend the letters used refer to the similar parts of the four blast furnaces (Figs. 160 to 196) chosen as characteristic types.

LEGEND.

A. Cast-iron breast jacket, right.
B. Cast-iron breast jacket, left.
C. Shaft, red brick shell.
C'. Shaft, firebrick lining.
E. Cast-iron side jacket.
F. Wrought-iron side jacket.
G. Wrought-iron back jacket.
H. Wrought-iron front jacket.
a. Slag-spout.
b. Crucible.
c. Lead-well or basin of Arents' automatic tap.
d. Syphon, the inclined channel of Arents' automatic tap.
e. Water jackets.
f. Cast-iron water feeder.
g. Lateral water-supply pipe.
h. Water-feed pipes.
i. Water overflow pipes.
j. Galvanized-iron water trough.
k. Cast-iron drain pipe.
l. Breast of furnace.
m. Tap-hole.
n. Tapping jacket.
o. Tuyere.
p. Blast-pipe.
q. Wind-bag.
r. Bustle-pipe.
s. Induction-pipe.
t. Cast-iron collar, supporting—or carrier-plate.
u. Cast-iron pillar.
v. Down-comer.
w. Sheet-iron curtain.
x. Main water-supply pipe.
y. Lugs fastened by bolts.
z. Sheet-iron hood.
a'. Lead-spout.
b'. Wrought-iron bolts.
c'. Corner-irons for tie-rods.
d'. Tie-rods.
e'. Interior of water jackets.
f'. Channel of water feeder.
g'. Brick arch.
h'. Crucible castings.
i'. Strengthening-ribs of crucible-castings.

*Devereux: 1887, December 6, No. 374,239; 1888, April 17, Nos. 381,118 and 381,119; June 12, No. 384,349; 1889, July 23, Nos. 407,335, 407,336, and 407,337; December 17, Nos. 417,314 and 417,315; 1890, May 6, No. 427,058. Konemann: 1888, October 9, No. 390,785. Wilson: 1889, May 21, No. 403,815, and others.

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j'. Telescope stack.
k'. Chains.
l'. Counter-weights.
m'. Feed-door.
n' I-beams.
o'. Capital of pillar.
p'. Brass nozzle.
q'. Hand-hole.
r'. Wrought-iron pipe connecting the single jackets.
s'. Eye or peep-hole.
t'. Cast-iron flange.
u'. Hog-chain.
v'. Iron band.
w'. Top-plate.
x'. Pipe leading to fan.
y'. Bed-plate.
z'. Hood leading into pipe *x'*.
a''. Tuyere-box.
b''. Steel rails.
c''. Wrought-iron rods.
d''. Expansion-space.
e''. Sliding sheet-iron door.
f''. Angle-iron ring.
g''. Angle-iron damper.
h''. Crank with nut.
i''. Circular guide with slot.
j''. Groove for damper.
k''. Water-feeder.
l''. Peep and poking-hole.
m''. Outlet for furnace gases.
n''. Swinging valve.
o''. Cross-pin.
p''. Movable weight.

Figs. 160 to 164 represent the blast furnace of the Omaha and Grant Smelting and Refining Co.'s Works at Denver, Colo., in 1883; Figs. 165 to 171, the furnace of the Globe Smelting and Refining Co., at Denver, Colo., 1891, designed by Iles; Figs. 172 to 174, a furnace from the Colorado Iron Works at Denver, 1891; Figs. 175 to 196, the furnace of the Montana Smelting Co., Great Falls, Mont., 1891, designed by Eilers.

While the general features of these furnaces remain the same, thus justifying the retention of the older illustrations, some important changes have been made, the leading ones being an increase of distance between the tuyeres and of height of the furnace. Other changes and improvements in construction will be represented in other illustrations.

No drawing of the circular furnace is given, as it is little used now; its advantages and disadvantages will, however, be discussed in connection with the oblong furnaces.

The materials required for the erection of a furnace, as shown in Figs. 165 to 171, are: cast iron, 27,300 lb.; wrought iron, 3,200 lb.; steel beams, 4,250 lb.; firebrick, 9,500; red brick, 17,000. Where a "telescope" stack is used, 1,600 lb. of wrought iron must be added to the above figures. The cost of erecting the furnace at Denver, excluding all the fittings for blast and water, was about $1,200, one-quarter of which went for labor.

Foundation.—The first thing in erecting a furnace is to have a solid foundation. Its depth will depend on the character of the subjacent ground. If bed rock is exposed this will furnish as good a foundation as can be wished for. If there is loose soil or gravel covering bed rock for not more than 10 ft., it is best to excavate until this is reached; otherwise a depth of 5 ft. will usually be sufficient to start the masonry below frost line, and to

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FIG. 160.

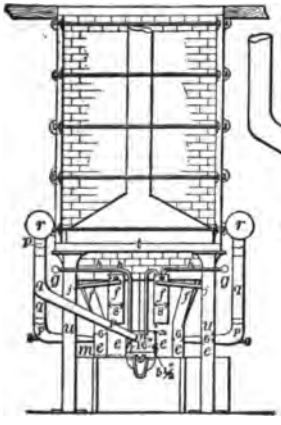


FIG. 161.

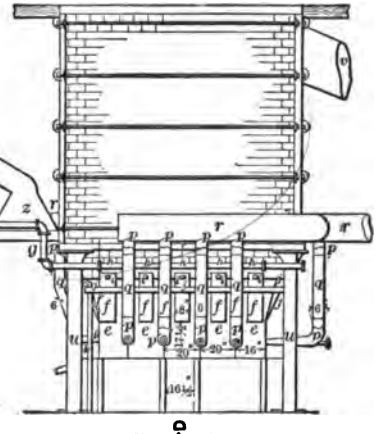


FIG. 162.

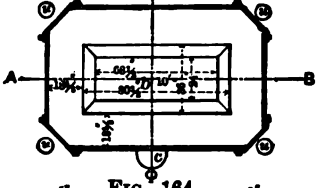


FIG. 163.

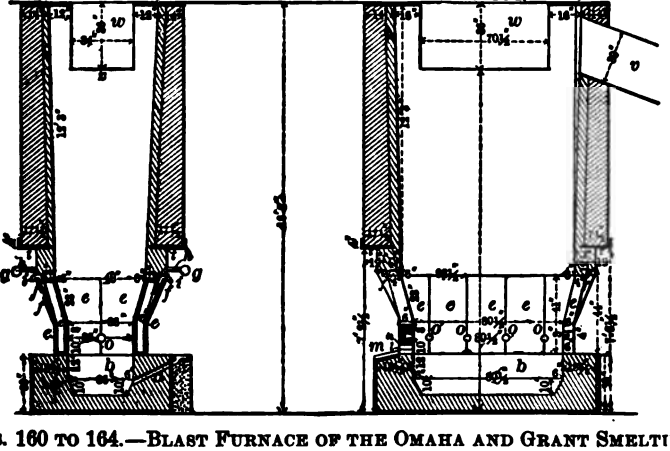


FIG. 164.

FIGS. 160 TO 164.—BLAST FURNACE OF THE OMAHA AND GRANT SMELTING AND REFINING CO., DENVER, COLO.

give the foundation the requisite strength. With very loose soil it is sometimes advisable to place in the bottom of the pit two layers of 3 or 4-in. planks spiked crosswise to each other, and

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upon that to build the foundation, which should extend from 2 to 3 ft. beyond the bed plate and the four pillars. It is built up of undressed rock, well rammed into place, the largest pieces being used for the corners, and care being taken to fill up the crevices and joints with as many spalls as possible; the whole is well grouted with a mixture of four parts of lime mortar and one part of cement. The topmost course must be absolutely smooth and horizontal, being generally of brick.

If one furnace is already in operation and a second one is to be erected, the simplest way of obtaining a good foundation is to empty the liquid slag into the place that has been excavated, or to throw in broken-up slag and to cement it with liquid slag. The top is evened and leveled by making shallow rectangular areas surrounded by sand or pieces of iron rails, and filling them level with liquid slag. Any ridges or other rough parts that remain are removed by chipping.

On the foundation is spread a thin clay mortar, upon which the wrought-iron bed plate *y'* (Figs. 165, 166, 172, 174, 175, 177) is placed.

Shaft.—On the foundation are erected the four hollow cast-iron pillars *u* (Figs. 160–162, 165, 166, 168–170, 172–177) which are to support the shaft.

The *height* of the shaft, *i.e.*, the distance from the center of the tuyeres to the feed floor, has been somewhat increased of late. It used to be from 10 to 12 ft.; it was then increased to 14 ft., later to 16 ft., which is the common dimension now, although occasionally it reaches 20 ft. The increase of height has been necessitated by the greater pressure of blast required for the highly silicious and calcareous slags. The ferruginous slags, formerly made, needed only a pressure of from 8 to 10 oz. To-day the pressure ranges from 1¼ to 3 lb. This increase of pressure has been made necessary by the enlargement of the distance between the tuyeres to increase the capacity of the furnace, and has been found to greatly increase the smelting power. That this is not an unmixed blessing will be shown later on.

The *horizontal section* of the shaft is either a circle or an oblong. When a square or polygonal furnace is blown out, the inside will have a similar appearance to that of the circular, as the corners very soon fill up. The circular furnace gives, as regards the

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FIG. 167.

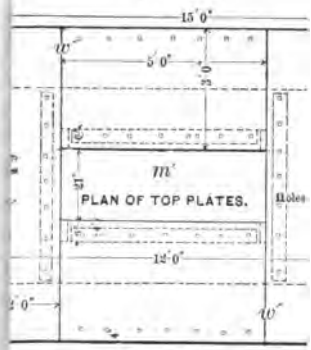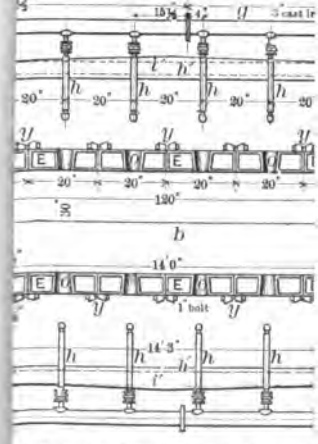


FIG. 168.



N THROUGH WATER JACKE

REFINING CO., DENVER, CO

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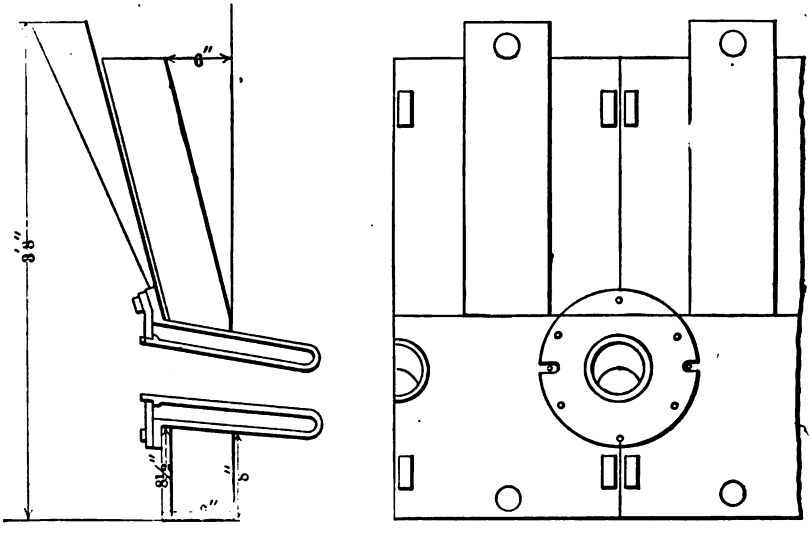
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quality of work, satisfactory results; there is an even distribution of blast and heat, and as it offers the largest surface for the smallest circumference, the loss of heat by radiation is the least possible. The drawback lies in the quantity of the work, which is limited, since the diameter at the tuyere-section must not exceed 48 in. (42 in. being the common dimension), as too great a pressure of blast would be required. For large quantities of ore the oblong form is therefore the proper one, as the area can be enlarged by making the furnace longer without increasing the distance between the tuyeres. Thus the length of some oblong furnaces has been doubled in the last 10 years, from 60 to 120 in.; some furnaces are even 140 in. long. In increasing the width the pressure of the blast has to grow, and with it the height of the furnace. With high-pressure blast, using the same percentage of fuel as with a low-pressure blast, the heat creeps up and smelting begins at the top of the jackets, in which case the walls are liable to be eaten out, instead of just above the region of the tuyeres, which of course is liable to cool and to cause rich slags and rich mattes. Another result is a greater loss of metal, as the retarding effect of the boshes on the hot gases becomes weakened and the working height of the furnace diminished. It has been found, however, that within limits an increase of blast permits working with a smaller percentage of fuel. The furnace putting through more charges with less fuel keeps cool at the top; nevertheless the fact remains that the mattes are somewhat higher in lead to-day than they were three or four years ago, if special precautions are not taken to liberate the lead, *e.g.*, by the addition of scrap iron. Furnaces have been built 48 in. wide at the tuyeres, the water-cooled nozzles protruding 6 in. into the furnace, thus making the distance between the jackets 60 in, as shown in Figs. 197 and 198. Such a furnace, 120 in. long and 20 ft. high from tuyeres to feed floor, with six 4-in. tuyeres on either side and a blast-pressure of 2 lb. is putting through 200 tons medium open charge containing 10% lead and 3.5% matte with 10% coke, the slags made being SiO_2 36%, FeO 32 to 33%, CaO 18%; and SiO_2 36%, FeO 28 to 29%, CaO 20%. This is an exceptional case. The furnaces usually built to-day are 42 by 120 in. at the tuyeres and have a working height of from 15 to 17 ft.; they have six 3½ to 4-in. tuyeres on either side and put through, with from 1¼ to 2 lb. blast-pressure and 12% fuel, from 85 to

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100 tons of medium coarse charge containing about 12% lead and from 3 to 5% matte.

The *vertical section* of all oblong furnaces shows the bosh, and with it the contracted tuyere section. This last secures a more perfect and rapid combustion, and thus a concentrated, intensified heat, with the result of a quicker fusion and a more complete decomposition of sulphide and arsenide of lead. If somewhat higher up the width of the furnace is suddenly enlarged by the bosh, the zone of fusion will be narrowed; further, the gases generated at the tuyeres will be evenly diffused, thus checking the velocity of their upward motion; by gradually giving up



FIGS. 197 AND 198.—WATER-COOLED TUYERE NOZZLE.

their heat they prepare the charges for the subsequent smelting process and decrease the amount of fluedust formed.

That a circular furnace having the form of an inverted cone cannot fully possess the same advantages is evident, but whether the enlargement between the tuyeres and the throat of the furnace be sudden or gradual, the relative areas of the hearth and of the throat remain nearly the same—1 : 2 or 2½.

The shaft (c.c. Figs. 166, 174–176, 178) is made of common brick c, and lined with firebrick c′ up to the feed floor. Lately in some furnaces an expansion space has been left open between

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the two walls, as is customary with iron blast furnaces. Only in very exceptional cases will the shaft consist of wrought-iron water jackets.* It rests on four supporting plates *t*, which used to be (Figs. 160, 161, 163, 164, 174) fastened to the capitals *o'*, of the pillars *u*, but the effect was to loosen the pillars by the unequal expansion of brickwork and cast-iron plates, and thus endanger the safety of the shaft. To relieve the pressure upon the plates, brick arches (*g'*, Fig. 172) were introduced, supporting the walls of the shaft and throwing the weight upon the pillars. To counteract the lateral thrust the lower part of the shaft is inclosed in cast-iron plates (Figs. 160, 161, 172, 174) firmly bolted together, in addition to which the plates *t* (Fig. 174) sometimes have flanges *t'*. Later a set of three I-beams (*n'*, Figs. 165, 166, 169, 175, 176) on each side of the furnace formed the support of the cast-iron plates *t* (Figs. 175, 176), being firmly bolted to each other and screwed tightly to the capitals *o'* of the pillars. The cast-iron plates *t* were in no way fastened to the beams, but rested freely upon them. By this arrangement supporting plates and shaft could expand independently of one another without endangering the stability of the shaft. The plates, being supported by the I-beams, no longer needed to be reinforced by the arches, but could bear the weight of the shaft safely. The disadvantage of both arrangements is that it is not easy to protect the plates and I-beams at the sides from destruction by heat when the furnace lining burns away, which is more frequently the case now with the high-pressure blast than it used to be. Beside, they are in the way when the lining is to be repaired or replaced. These difficulties are obviated by the use of arched steel girders (Fig. 199)† which carry the red-brick walls, their lateral thrust being taken up by tie-rods, while the mantle or supporting plates at the ends are retained. Inside of the girders are angle bars and plates which carry the firebrick lining. When this burns thin the angle bars are liable to become warped, but as they are independent of the arched girders, it does not affect the rest of the furnace. In order to make the burning out of the lining impossible, water-jacketed girders (Fig. 200)† carried by the columns have in some instances taken

* Examples: "Transactions of American Institute of Mining Engineers," xxi., p. 575; *Engineering and Mining Journal*, Nov. 18, 1893.

† Taken from the catalogue of the Colorado Iron Works, Denver, Colo.

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IGS. 199 AND 200.—GIRDERS FOR SUPPORT OF BLAST FURNACE SHAFT.

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the place of the lower firebrick lining. A suggestion for solving the difficulty in a very simple way is made by G. Murray.* It is to run an I-beam through the furnace-wall, 5 ft. or more above the place where the mantle or supporting plate is usually located. It would carry the upper part of the shaft, and from it would be suspended by bolts the supporting plate, which carries the lower part, thus leaving freely exposed the space above the jackets that is liable to burn out, so that it may be easily repaired.

The walls of modern furnaces are made very thick at the bottom in comparison with those of earlier ones (32½ to 39 in. against 17½ in., Figs. 165, 166, 175, 176 against 164, 165), decreasing toward the feed floor, which causes a considerable saving of fuel.

The entire shaft is well braced with tie-rods *d'*, secured in corner-irons *c'* (Figs. 160, 161, 165, 166, 171, 172). Austin lays old rails in the brick every 12 or 24 in., more frequently at the bottom than at the top, in such a way that the foot shall be flush with the outside. Each corner is covered with 4-in. angles and bound by tie-rods 2 ft. apart.

Feed Holes and the Collecting of Fumes.†—There are two general arrangements for feeding by hand and carrying off fumes. The first is to have a feedhole on either side of the furnace *m'* (Fig. 178). The stack or chimney *c* (Figs. 178, 179) is of brick, forming the continuation of the shaft; it is contracted at the upper end to about 3½ ft. square (inside measurement), and then passes through the roof. The top is closed by a swinging damper resting in the groove *j''*, and can be opened from the feed floor by means of a damper rod.

In small furnaces the feed holes are placed in the middle of the sides; in large ones near the front and back of the furnace, diagonally opposite each other, or they run nearly the whole length of the furnace. This makes it much easier to distribute the charge evenly and to bar down the wall accretions. The doors (Figs. 184, 185, 186) are 5 or 6 ft. high, in order that a man may be able to stand in them and direct the bar. During the run they are closed to from 18 to 24 in. by letting down a sheet-iron curtain *e''* that slides in a cast-iron frame (Fig. 184) and is balanced by a counter-weight. In order to prevent the door from becoming wedged fast in the frame, it is advisable to

* Letter, March, 1897.

† Hering, *Dingler's Polytechnisches Journal*, 1886, cclxi., p. 205.

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suspend it not by a single rope, as shown in Fig. 185, but by two ropes attached to the ends and passing over sheaves keyed to one horizontal shaft. In whatever way the door may be raised or lowered it will thus always swing true. The sill of the feed floor is often a foot or less above the floor level, so that the feeder cannot simply shovel the charges into the furnace, but, being obliged to raise the shovel each time, will be more likely to distribute the charge evenly.

The fumes are drawn off near the top of the stack by a circular sheet-iron flue *v*, which passes at a steep angle into the dust chamber. In the flue is the damper (Figs. 178, 180, 181, 182) to regulate the draft, which is kept just strong enough to prevent the gases from passing out through the feed holes. In blowing in or blowing out, the damper in the flue is closed and the one on the top of the stack thrown open.

This method has the advantage over others that less dust is carried off into the dust chambers, as the speed of the gases at the top of the charge is lessened by air drawn in through the feed doors, and the mixture having to rise in the shaft, much dust drops back on the charge.

The other arrangement is to feed the furnace from the top (Figs. 163–167, 172–174). This is covered in part by cast-iron plates *w'* (Figs. 167, 174), leaving an opening through which the charge is introduced. Formerly the Pfort method (Figs. 163, 164) of carrying off the gases was in general use. It consists in suspending from the cast-iron top-plates *w'* an iron curtain *w*, so as to leave room between it and the walls of the furnace for the gases, whence they pass off through a flue *v* into the dust chamber (Fig. 156), the charge filling the inside of the curtain up to the feed floor. While this arrangement proved very satisfactory in a good many ways, it had two striking disadvantages: first, that it was impossible to see how the charge sunk except by dropping it below the edge of the thimble; and second, that the time required for barring down wall accretions was lengthened, because it is necessary to remove the curtain before and to put it back again after barring. For this reason the curtain has been almost universally discarded and the gases are simply drawn off by a flue *v* (Figs. 172, 174) at the back of the furnace, or better by two on the sides. To avoid sucking in air or letting out fumes, the feed opening *m'* in the cast-iron top-plate *w'* (Figs.

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166, 167, 174) is made rather small; it is, however, large enough for the feeder to be able to spread his charge in any way that may be necessary, and to reach any part of the side walls with a bar when cutting out wall accretions. The flue leading to the dust chamber, which used to be of sheet iron (Fig. 164), is now commonly built of brick (Fig. 172). It rests on heavy rails and is thoroughly bound with buckstays and tie-rods.

Formerly a "telescope stack" (*z*, Fig. 156, and *j'*, Figs. 172 and 174) was suspended by chains *k'* over each furnace. It is a sheet-iron pipe reaching through the roof and balanced by counter-weights *l'*. Its lower part is enlarged to the oblong form of the feed opening and has a small feed-door *m'* on either side. The stack is lowered when the furnace smoke is not sufficiently drawn off by the flue, or when the furnace is blown in or out, to carry off the gases into the open air.

At present, large smelting plants have one or more of these sheet-iron stacks suspended from a traveler, to be used in case of necessity, and in some instances this stack has been thrown off entirely (Figs. 165 to 171). Instead, above each furnace is suspended a ¾-in. cast-iron plate sufficiently large to close the feed opening *m'*. It is lowered when the furnace is being blown out, to prevent the fumes from passing to the feed floor; the joint is made air-tight by spreading moistened fine ore over it.

Another manner of collecting the waste gases with open-mouthed furnaces may be mentioned here. This is the Darby tube, a wrought-iron pipe of small area in comparison with that of the throat of the furnace (280 sq. in. as against 3,267). It is hung in the middle from the girders, some distance (5 ft. 2½ in.) down into the furnace. This tube is used in the Upper Harz Mountains, where fine galena concentrates are smelted raw. The advantages claimed for it are that the charge is less liable to pack toward the center, and that the gases, being drawn off there, are prevented from rushing up at the sides, and penetrate the charge more evenly.

The idea of closing the throat of the furnace with cup and cone and feeding automatically has been often suggested, *e.g.*, by Hahn,* but the experiments have not proved successful, because the heat crept up in the furnace and gave a hot top. The only successful mechanical feeding, as far as the writer is aware, is

* "Mineral Resources of the United States," 1882, p. 343.

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that found at the works of the Pueblo Smelting and Refining Co. (See § 66.)

Hearth with Arents' Automatic Tap.—The bottom of the hearth is formed by a bed plate of boiler iron *y'* (Figs. 165, 166, 172, 754–176), which is to prevent any lead from percolating downward. It is placed on the foundation, as indicated in § 67. Care must be taken to have its center coincide with that of the shaft by dropping a plumb-line from the feed floor. The bed plate sometimes has an angle-iron rim inclosing the bottom course of brick (Figs. 165, 166, 175, 176); sometimes it reaches beyond the castings (Figs. 172 and 174), inclosing the hearth, which rests upon it.

These castings *h'* (Figs. 162, 165, 166, 168, 172–177), reaching to the top of the hearth, have been (and still are) a great trouble, as they are very liable to crack. At first they were made 1 in. thick; later the front and back plates were strengthened by ribs *i* (Figs. 165, 166, 168, 172–177); then the side plates, castings, and ribs have been made thicker and the beveled corners have been fastened together by special tie-rods *b'* (Figs. 165, 168). Still there is danger of their cracking, so that in some furnaces a wrought-iron band *v'* (Fig. 166) is screwed to the sides to hold the casting together. It would seem as if making the outer wall of the hearth oval and inclosing it with a $\frac{3}{8}$-in. wrought-iron plate, as is done in the modern large iron blast furnaces, would be the way to solve the difficulty.

If the furnace has an internal crucible, as is commonly the case, the slag spout *a* (Figs. 165, 167, 172, 173, 175, 177), which is sometimes water-jacketed, is fastened with bolts to the casting on the front, as is the lead spout *a'* (Figs. 174, 176, 177), if the well is confined within the hearth plates, as is now usual. It is not often that the slag is tapped alternately from the front and the back of the furnace, requiring two slag spouts (Figs. 165, 168, 172, 173), and that the lead is removed from the two sides (Fig. 174). Two slag taps have been used at some furnaces to counteract the forming of a crust at the back of the furnace, where it usually begins, growing toward the front, from which it cannot easily be reached. If the slag is tapped from both front and back, the danger of crusting at the back is in part at least avoided, and if an obstruction forms there, it is easily removed. Another advantage, at least with a furnace putting through a large amount of charge,

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is* that it will lessen the hurry and inconvenience of wheeling to and from the tap-hole a large number of slag-pots which in any case is a hot and disagreeable piece of work, and is made more unpleasant by the spattering of melted materials. Having two lead-wells is simply a waste of heat, for when one becomes clogged up, the other will also.

At some works a crucible that is partly external and partly internal is used in connection with the Arents' siphon tap. Two examples may be given. Eurich† has replaced the ordinary tymp stone by a dam-jacket consisting of a zigzag wrought-iron pipe with a coating of cast iron. It is shorter than the width of the furnace, and has on either side a half-course of brick, thus facilitating its putting in place and removal. The forehearth, reaching to the level of the tuyeres, is an oblong casting lined with firebrick. The slag overflows continuously, and the matte is tapped periodically. Mathewson‡ constructed a forehearth that can be readily attached to any existing furnace, the only change necessary being a new set of front jackets. It does satisfactory work if the furnace puts through not less than 50 tons of charge in 24 hours. Its latest form is shown in Figs. 201 to 206.

"It consists of a detachable cast-iron forehearth *G*, from which the clean slag overflows continuously, and a matte tap *E'* placed at the usual level. A water-cooled tymp K closing the upper breast of the furnace is fastened by bolts to the front jackets *B*. The detachable forehearth *G* consists of two water-cooled cast-iron sides, joined at the lower ends by the water-cooled matte-tap *E*, and a front plate *H*, usually not water-cooled. Sides and matte-tap are held in place at the bottom by the front wall of the crucible *A*, and at the top by the two key-bolts; the front-plate is attached to the sides by lugs. The slag, which separates from the matte in the furnace, enters the forehearth through the opening E^2, and overflows on the side through the opening G^2 into the slag-spout *J*, which can be raised and lowered, as desired, in the guides *G'*. The matte is tapped at intervals in the usual way."

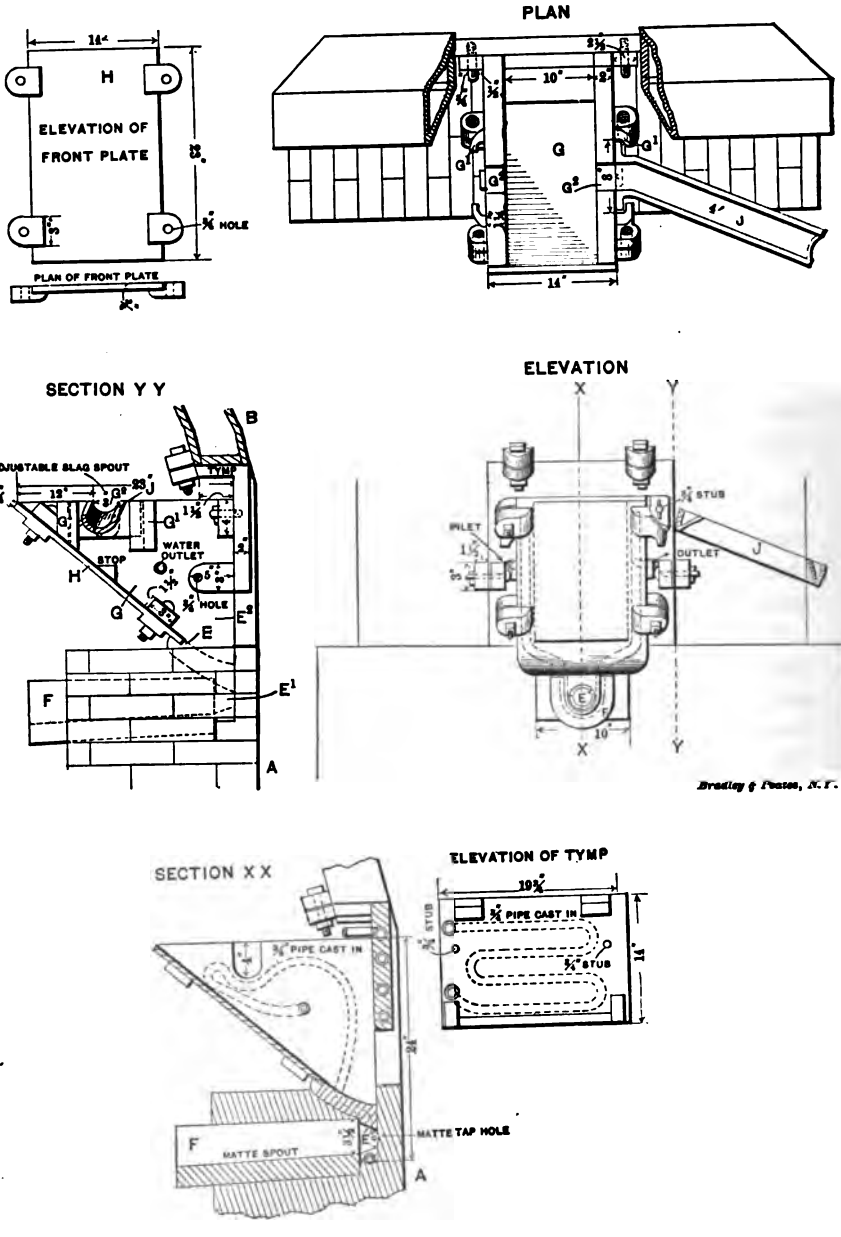
The forehearth weighs 300 lb. In order to insure the slag from becoming accidentally rich, when the rod is used or from

* Henrich, "Transactions of American Institute of Mining Engineers," xxv., p. 96.

† U. S. Patent No. 494,104, March 25, 1890; private notes, 1896.

‡ U. S. Patents No. 488,986, Oct. 4, 1892; No. 501,351, July 11, 1893; No. 515,244, Feb. 20, 1894; Hofman, "The Mineral Industry," ii., p. 435; iii., p. 435; private notes, 1896.

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FIGS. 201 TO 206.—MATHEWSON'S IMPROVED FURNACE TAP.

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some other cause, it is advisable to run it into an overflow-pot. In a 200-ton furnace having 3.5% matte on the charge, the overflow-pot should be replaced but once in six hours.

The hearth walls and bottom of a crucible are of firebrick. They usually rest on the bed plate (Figs. 163, 164, 166, 174). Sometimes on the bed plate and below the crucible a 6-in. layer of ground brick and raw clay (3 : 2, by volume) is beaten down firmly in the form of an inverted arch (Figs. 175, 176) on which the bricks forming the bottom proper are placed. In building the side walls it is better not to place the bricks in direct contact with the castings (Figs. 163, 164, 174–176), but to leave a small space (about 2½ in.) and tamp it out with brasque (equal volumes of ground coke and clay) while the bricks are being placed (Fig. 166). In this way the crucible, when it expands, will simply pack the brasque tighter, and thus relieve the castings from at least part of the strain.

In order to prevent any lead from leaking out at the front through the brickwork, and being carried off with the slag, a water-block with trough-shaped slag-gutter is often built into the brickwork. If any labor has to be performed in the crucible, a free working opening is easily made by removing the tapping jacket and water-block.

Arents' automatic tap, or siphon tap, which forms part of the side wall, consists of an inclined channel, the siphon *d* (Figs. 163, 166, 174, 176), 3 or 4 in. square, running from the lowest part of the crucible wall inside to the top on the outside, where it is enlarged into a dish-shaped basin—the lead-well *c* (Figs. 163, 166, 173, 174, 176, 177), the length, of course, depending on the depth of the crucible, which varies from 22 to 30 in. The tap is usually in the middle of one of the sides (Figs. 172, 173), between a pair of tuyeres, although sometimes placed nearer the front (Fig. 177); while the furnace is running the crucible remains nearly full of lead, that in the automatic tap standing higher on account of the pressure of the blast. With the high pressure in present use, it often reaches to nearly the level of the tuyeres. From the well the lead, as fast as it is made in the furnace, is either ladled at once into molds, or goes first into the cooling-pot, into which it overflows, or more commonly, is tapped at intervals. When first used,* Arents' tap consisted of

* Hahn, Eilers, Raymond, "Transactions of American Institute of Mining Engineers," i., p. 108.

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a 3-in. wrought-iron pipe which terminated a foot or more below the upper rim of a sheet-iron cylindrical shell rammed full of fireclay and bolted to the casting, the well being afterward cut out. The wrought-iron pipe has been universally abandoned. The remains of the sheet-iron cylinder are still found in some instances in a half-cylinder that is bolted to the casting (Figs. 162, 163). In most furnaces to-day the lead-well is inclosed in the crucible wall (Figs. 166, 172, 174, 176, 177). The advantages of this improvement are that the siphon is shortened and the lead is kept hotter. In smelting charges that run high in lead, the lead in the crucible is frequently exchanged, and a slight loss in heat does not make itself felt; the siphon can be long, and the well, not being close to the tuyere-pipes, will leave these cool. With charges low in lead, all loss of heat must be avoided. The increased thickness of the side wall, necessitated by the inclosed lead-well, is altogether an advantage, because the loss of heat is diminished. In any case the side walls ought never to be less than 22-in. thick.

The advantages of the automatic tap are many. Without it the lead and matte are tapped from the bottom of the crucible, and to do this the blast is shut off, the blast-pipes are removed, and the tap-hole is opened, which is often done with difficulty. Lead and matte run out into a shallow tapping kettle, and the moment the slag appears the opening must be closed with a stopper of brasque or clay. Then the crucible is cleared by inserting through the forehearth (if it has one) a curved iron bar, the tuyeres are cleaned with iron rods, the blast-pipes are put in place, the blast is then turned on again slowly, and smelting is resumed. This stoppage takes considerable time, and therefore cools the furnace. Then into the crucible (now free from lead and matte, although it retains some fuel) falls an equivalent amount of half-melted charge, which has gradually to be lifted up by fresh lead and matte, when tapping begins again. These half-melted masses thus have a chance to adhere to the bottom of the crucible, and are very apt to be the beginning of a bottom crust. If this has once started, it is nearly sure to grow and gradually freeze up the furnace.

With Arents' tap there is no stoppage of the furnace when the lead is removed from the lead-well or when matte and speise are tapped out with the slag, which is either into a slag-pot, where

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they settle out according to their specific gravities, or alone, the slag running off continuously. The furnace therefore runs more regularly, and the first formation of slag or matte accretions at the bottom is prevented, as the crucible is always filled with lead.

The claim originally made that the lead from the automatic tap is purer than that from the tapping kettle will hardly be maintained to-day. It was based on the theory that the metal, being taken continuously in small quantities from the bottom of the crucible, would be purer, as the heaviest, *i.e.*, the purest lead, would gather there, and the impurities would float to the surface, to be taken up by matte and slag. This presupposes that the lead in the crucible is sufficiently undisturbed to permit liquation. The fact is that a constant current in the lead prevents this separation of dross and lead. The dross is disseminated through the lead; some reaches the surface and is taken up by matte and slag; but a large part of it rises in the siphon and collects in the well. Thus the bars from the lead well are often less pure than those from the tapping-hearth, as with the latter the dross adheres to the cake of matte floating on the still liquid lead. However, by skimming the dross from the well or cooling-pot, clean bars can be obtained, and the dross returned at once to the charge, thus involving little loss of metal. With the tapping-hearth, the dross adhering to the matte must undergo all the operations with it, and thus much lead and silver are lost. Thus the automatic tap insures a considerable saving in metal. At some works, in order to lessen the large amount of lead lying idle in the crucible, the space has been diminished by partially bricking it up at the back.

There is one case where tapping from the bottom is to be preferred to the automatic tap. It is in smelting coppery ores—as an alloy of lead and copper separates out from the lead, adheres to the bottom of the crucible, and grows upward, filling it (see § 67). The trouble is remedied by adding sufficient sulphur in some form or other to form matte. As soon as this runs 12% copper the difficulty begins again to make itself felt. In concentrating lead matte with 12% copper or more, the automatic tap is out of place, and the ordinary copper furnace with internal crucible is used, or, as will be seen later, the lead furnace can be worked in the same way as the copper furnace.

Water Jackets.—These (*E*, Figs. 165, 166, 168, 172, 174) are

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water-cooled iron shells that inclose the smelting zone of the furnace to protect it from the corrosion of the slag. Since about 1873 they have come into more general use, and have now entirely replaced the brick walls at the region of the tuyeres, whenever there is sufficient water to warrant their use. There is no doubt that the continuous flow of water through them abstracts a considerable amount of heat, as has been lately again emphasized by Lang,* but their use has so many advantages† that sandstone, firebrick, or other refractory material is seldom reverted to unless the supply of water is insufficient.

Quite a discussion arose in 1885-86‡ as to the date of the invention of the water-jacket furnace and the inventor. The writer has failed to find any reference to water jackets in the treatises of Karsten§ and Scheerer.‖ The earliest mention of their use is made by Overman,¶ who describes and illustrates a refinery furnace, the sides of which consisted of water-cooled cast-iron shells, through which water-cooled tuyere nozzles protruded into the furnace. Douglas** says that J. Williams built near Drontheim, Norway, in 1852 "sectional water-jacket furnaces consisting of a circle of long, narrow water-backs, perforated by tuyere holes." About the year 1865 the same J. Williams erected a number of water-jacket blast furnaces at Houghton, Lake Superior. According to Arents,†† Haskell built in 1865 a water-jacket furnace in California. Kerl,‡‡ in describing the improvements made in smelting in the Harz Mountains, records the introduction in 1864 of "water-blocks to cool the hearth and to serve as a support for the water-cooled tuyere nozzles," but these had been used in refinery furnaces for a very long time,§§ and are not to be confounded with water jackets. The latter never were and are not to-day in use in the Harz Mountains.

* *Engineering and Mining Journal*, 1897, Jan. 3, Feb. 6, 13, March 13.

† Zwalmvenburg, *ibid.*, April 10, 1897.

‡ *Engineering and Mining Journal*, 1885, July 25 (Harnickel, Rolker); Aug. 1 (Courtis); Aug. 15 and 29 (Kleinschmidt); Aug. 22 (Editor); Sept. 12 and 26 (Williams), Oct. 10 (Hahn); Oct. 24 (Arents); Oct. 31 (Douglas); Nov. 7 (Courtis, Daggett); Nov. 14 (Curtis); Nov. 29 (Kleinschmidt); 1886, Jan. 2 (Tew).

§ "System der Metallurgie," Berlin, 1832; and "Handbuch der Eisenhüttenkunde," Berlin, 1841.

‖ "Lehrbuch der Metallurgie," Brunswick, 1846-53.

¶ "Treatise on Metallurgy," New York, 1852, p. 556.

** "Mineral Resources of the United States," 1882, p. 268.

†† *Berg- und Hüttenmännische Zeitung*, 1866, p. 316.

‡‡ *Ibid.*, 1867, pp. 6 and 47.

§§ Percy, "Metallurgy of Iron and Steel," London, 1864, pp. 584 and 625.

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Courtis, who made a drawing of the Pilz furnace at Freiberg in 1866, says that the tuyere and the front of the furnace were water-cooled. Water jackets have been introduced at Freiberg and Pribram only since they became common in this country; they were used in the Saint Louis Smelting Works near Marseilles before 1878.* Spray jackets were used at La Pise as early as 1862.†

The water-cooled tuyere nozzles, which resist the action of heat and slag so well, while the brick walls are eaten out, appear to have suggested to different persons the idea of extending the water-cooled iron surface, and thus caused the construction of the water jacket. This would give several men the credit of having invented it.

The water jackets (*e*, Figs. 163, 164; *e'*, Fig. 166; *E*, Fig. 174; *F*, 176; *H*, Fig. 175) are placed on top of the hearth walls, forming their continuation on the inside. Their height has varied from 2 to 4 ft.; 3 ft. 6 in. is an ordinary measure. They thus reach from the top of the hearth to within about 12 in. of the cast-iron carrier plate *t*, or the I-beams *n'* which used always to support the shaft. The center of the tuyeres is placed 10 in. higher than the bottom of the jackets, and from 8 to 10 in. above this begins the bosh, the amount of which varies from 6 to 10 in. (occasionally 12 in.) in 30 in.; but the greater the amount of bosh the more liable are wall accretions to form there.

Jackets are made of cast iron and low-carbon steel.

The cast-iron jackets (*E*, Figs. 160, 161, 163-166, 172-174) are generally 6 in. thick, the sides being of $\frac{3}{4}$ or $\frac{5}{8}$-in. iron. Each jacket has its own water-feeder *f*, which begins 8 or 10 in. above the center of the tuyere and runs from 3 to 4 in. above the top of the jackets, extending outward about 4 in. This insures the complete filling of the jacket with water. As the top of the feeder is closed only by a lid, tools can be introduced to scrape off the scale. The feeder was formerly always cast in one piece with the jacket, but now it is often a separate casting, fastened on with screws or bolts. At first there was no opening at the lower end of the jacket (Figs. 160, 161, 172) to remove mud or scale that had collected; now there is usually a hand-hole (*q'*, Figs. 165, 166) for this purpose, and thus the life of the jacket is much

* Grüner, "Traité de métallurgie," Paris, 1873-78, vol., ii., p. 391.
† Grüner, *Annales des Mines*, 1868, xiii., p. 364.

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prolonged. The tuyeres (*o*, Figs. 160, 161) always used to be at the junction of two jackets, each having a semicircular recess. This was for fear that an opening through the center of the jacket might weaken it; but it has not proved to be the case, and the tuyeres are now (Figs. 165, 168, 172, 173) generally made in that way. Tuyeres between jackets are sure to cause considerable leakage of air. When a furnace is new, and the two semicircular recesses are just opposite each other, and a brass nozzle (*p'*, Fig. 174) is inserted, which receives the galvanized iron blast-pipe, this does not at once occur, but is sure to come later. Not less than six different kinds of jackets were formerly used in a large furnace. Now the number has been reduced to three (Figs. 168, 171), two kinds, *A* and *B*, on both front and back, and one, *F*, on the sides—or, if front—and back—jackets have a bosh, by making them like the side ones and having two forms of end-jackets with curved corners. To these must be added the tapping jacket (*n*, Figs. 160, 166, 175). This reduction in number is made possible by giving the front of the furnace the same construction as the back, the opening required at the front being simply bricked up at the back. The length of the front jackets varies somewhat in different furnaces. Sometimes they reach to within 10 in. of the top of the crucible (Figs. 175, 187), thus leaving open a 10-in. breast, which runs across the entire width of the furnace. It is usually closed by two small brick pillars (9 by 4½ in.) and three balls of clay, one in the middle and one on the outer side of each pillar. In the central clay-ball is placed the slag-tap. There is nothing in this arrangement to prevent a tapping jacket from being put in. A second plan is shown in Fig. 160. Here the front jackets reach at the sides down to the top of the crucible, and leave in the center an opening, the upper part of which is closed by a separate jacket *n*, and the lower, the breast *l*, by a ball of clay, in which is placed the slag-tap. This plan is not a common one. A third modification (Figs. 166, 174) is to fill the open place in front with a tapping jacket to within 2½ in. of the top. This space is left for convenience in taking out the tapping jacket, and is closed by brick. The jacket (*n*, Fig. 175), 26 by 14 in., and 3½ in. deep, has, 6½ in. above its lower edge, a tap-hole *m*, which is 2½ in. in diameter, and widens, after entering the jacket to the depth of 1 in., to 5 in. on the inner side. It was formerly made exclusively of cast

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iron; to-day cast steel is found in many works. The lower edge of the jacket is placed 4 in. beneath the upper edge of the crucible castings, and thus prevents at the front that leakage of lead from the crucible which is otherwise so difficult to stop. Another way of stopping this leakage is to have a water-block imbedded in the front wall and to place the tapping jacket flush with the top of the crucible. Having large ore-beds from which to make up the charges there is little probability of trouble in the furnace, and therefore a small opening closed by the tapping jacket is sufficient for all practical purposes. At small smelters, where the charges are changed often, it is probably better to have the front jackets 10 in. smaller than the side jackets, in order to give room for working in the crucible, should it prove necessary.

The water jackets have been joined in various ways by using wedges, screws, bands, etc. Now they are simply bolted to each other near the top and bottom, the bolts passing through lugs (*y*, Figs. 165, 168, 172, 173) cast in the jackets.

A cast-iron jacket,* if made of the best iron and cast in dry sand, will last with good water about three years, while in the beginning of the eighties, when cast in green sand, their life ranged only from three to four months. If a jacket should begin to crack, either because imperfect when received or from having been injured by barring, this can be readily stopped. The crack is moistened with oil,† then wiped off and rubbed with chalk, when the oil, previously taken up, will exude and show exactly the points where the crack ends. Then holes are bored at either end, a shallow mortise with receding sides is cut along its length, the holes are closed by driving in copper pins, and the mortise by calking with a stout copper wire.

The Colorado Iron Works, Denver, Colo.,‡ have brought out a water jacket which is divided by a partition into two chambers, the outer one being an air jacket through which the blast passes, taking up some of the waste heat, and carrying it back into the furnace. The same firm recommends an evaporating jacket which by requiring a smaller amount of water is intended to save a large portion of the heat usually carried off by the overflow-

* Vezin, "Transactions of American Institute of Mining Engineers," xxvi., p. 1095.
† *Engineering and Mining Journal*, Nov. 15, 1890.
‡ Circular, November, 1896.

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water. Practical use will be necessary to demonstrate the value of the improvements.

Low-carbon steel jackets have in a few instances replaced those of cast-iron. With circular furnaces they are exclusively used. They have no bosh, and are usually in two parts. They seldom have separate water-feeders, as the cast-iron jackets do, the water inlet pipe being usually near the bottom and the outlet pipe near the top. With this arrangement, it is important, if a complete filling of the jackets with water is to be made possible, to have one or two small pieces of pipe protrude upward and outward from the top of the jackets, and to have the water outlet pipe also bent upward that it may discharge above the top of the jackets.

Oblong furnaces have, as a rule, only four steel jackets, one on each side. Occasionally they have been made in narrow sections, 20 in. wide, like the cast-iron ones. The disadvantage of large sections is that the walls must be stiffened by stay-bolts riveted on the fire-side, while with narrow sections this is not necessary; large sections are further liable to become distorted and in case of leakage the furnace has to be blown down. Two kinds of jackets are in use. One is shown in Figs. 175 to 177, and 188 to 191. The jacket is of the usual height; the inner wall forms a straight line slanting outward; the outer wall has a greater slant; thus the water-space grows wider toward the top, where the water outlets *i* are riveted to the outer wall, the inlets being at *k''*. The walls are stiffened by stay-bolts. The mud accumulating in the jackets is removed through the hand-holes *q'*. The details of the tuyere-box *a''* are discussed further on. The other form of jacket is like the cast iron one. The walls are vertical and parallel from the base to the bosh, and then slant outward; cast-iron water-feeders are bolted to the outer wall. If cast-iron and steel jackets be compared, there is no question that on the whole the latter last longer than the former, especially if made in small sections. If the water that is to cool the jackets be muddy or hard and thus liable to form scale, a steel jacket will not last much longer than one of cast iron. It is sometimes said that cast-iron jackets require less water than wrought-iron ones. This remains to be proved. If made in small sections a set of steel jackets weighs about half as much as a cast-iron set and costs about three times

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as much; if in large sections, the cost is twice as much. In out-of-the-way places, where freight charges are very high and there is no foundry, it may be of advantage to use steel jackets; in a centrally located place, steel jackets are little used.

The inner walls of cast-iron jackets and matte pots become corroded and absorb silver. Thus, according to Iles,* a matte-pot, having been freed from all adhering material, assayed 35 oz. silver per ton, and a water jacket gave similar results. Hence with an old jacket or slag-pot, only the non-corroded parts can be sent to the foundry.

The cooling water for the jackets is drawn from a wooden tank, the bottom of which should be at some distance above the water inlet, in order that there may be some pressure, as an extra amount of water is often needed in blowing in or blowing out. The main delivery pipe starts from the water tank and runs along the front or back of a row of furnaces. From it branch off separate supply pipes (x', Fig. 164,) each of which ends in a pipe g (Figs. 160, 164–166, 168, 172, 174, 176), surrounding its own furnace. This supplies the small feed pipes h, which deliver the water into the top of the feeders, the flow being regulated by a valve. The cold water, entering the jacket at the top, sinks down slowly and pushes upward the hot water, which runs off through the small pipe i, below the inlet. This, the common arrangement, accomplishes on the whole its purpose, but if the temperatures of a jacket be compared at the top and the bottom, it will be found that the bottom is always the hotter. There are two ways of equalizing the heat. One is to attach a rubber hose or loosely-fitted pipe to the feed pipe h, thus letting the cool water come in contact with the hot water at about the middle of the jacket. The other is to have an extra supply-pipe through which a small stream of water runs in near the bottom of the jacket. If these two methods do not succeed in cooling the lower part of the jacket, it shows that it contains mud or scale, and requires cleaning, if it is not soon to burn through. The hot water from the jackets is discharged into a galvanized-iron trough j (Figs. 160, 161, 172, 174) surrounding the furnace, from which it passes off through a cast-iron standpipe k (Figs. 161, 172) into a main underground. The troughs are often in the way when the furnace is running, and are very inconvenient sometimes, as, for

* Private notes, 1896.

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example, when a cracked jacket has to be exchanged. To remedy this, the water is sometimes carried away from the jackets by long pieces of gas-pipe, terminating in the funnels of standpipes placed close to the supporting pillars and connected with the

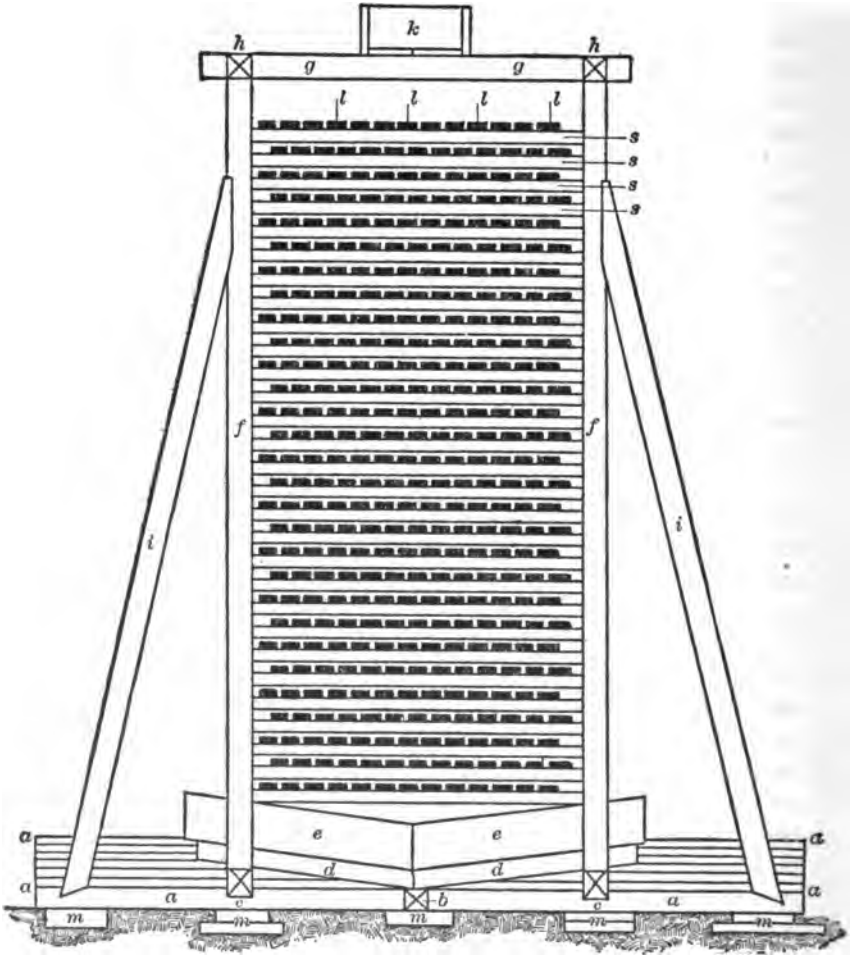


FIG. 207.—THE HENRICH WATER COOLER.

main underground, or there is a standpipe with funnel for each jacket or pair of jackets. Thus, if a jacket has to be exchanged, it is only necessary to turn off the water supply from it instead

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of from the whole side, as is ordinarily done, the trough being also removed. The amount of water required to cool the jackets varies with the size of the furnace and the slag that is being made. A furnace 36 by 120 in. at the tuyeres, making a silicious calcareous slag, requires under normal conditions 30 gal. of water per minute. This is a good average figure. For blowing in or blowing out one must be prepared to use double this amount. Part of the overflow from the jackets is often used as feed-water for the boilers. In many cases, where there is a lack of water supply, the same water has to be used over and over again in the jackets, which of course necessitates quick cooling. Henrich* has constructed a water-cooling apparatus entirely of wood for a Herreshoff elliptical water-jacket furnace in Tennessee, 42 by 120 in. at the tuyeres, which will cool per minute about 200 gal. of nearly boiling water down to the temperature of the air. The cooler (Fig. 207) is an oblong wooden structure, 18 ft. high, 8 ft. wide, and 48 ft. long, consisting of strips of wood (battens 1 by 4 in. and 16 ft. long(laid 2 in. apart and perfectly level, one set over the other 6 in. apart. The water pumped through a 4-in. pipe into the perforated distributing trough *K* (of 1½-in. planks), spreads over these strips and drops from one set on to the other, thus exposing a large surface to the cooling action of the air as the water passes slowly downward to be collected at the bottom *E*, whence it flows into the sump of the pump supplying the jackets.

The structure is erected on cross sills *a* (4 by 6 in. and 16 ft. long, put on edge) resting on mud-sills *m* (2 by 12 in.) firmly imbedded in the ground. The cross sills are 4 ft. apart and placed in steps 1 in. high, so that the last one, the thirteenth, shall be about 1 ft. higher than the first. The cross sills are joined in the center by a long sill *b*, halved into the cross sills, so that the tops will be flush, and on either side by a long sill *c*, let 2 in. into the cross sills. The long sills are of 6 by 6-in. lumber. At the intersection of cross sills *a* and long sills *c* are placed the posts, of 4 by 6-in. lumber and from 16 ft. 6 in. to 17 ft. 6 in. high, which are strengthened by braces *i* of 4 by 4-in. lumber. A pair of posts is joined by a cross cap *g* (6 by 6-in. lumber) and the 13 pairs of posts by long caps *h* of the same size. On the side, between the posts, are X-braces of 3 by 6-in.

* "Transactions of American Institute of Mining Engineers," xxv., pp. 43, 460.

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lumber to stiffen the structure. Strips of wood *s* (2 by 4 in. and 7 ft. long) are nailed 6 in. apart between each pair of posts; *e* is the flooring of $\frac{7}{8}$ by 4-in. lumber, and *d* the joists, 2 by 4 in. and 10 ft. long. In order to prevent the wind from carrying part of the water in the form of spray to one side beyond the floor, it may be necessary to put up a wind-break on one side. This is made of 1-in. boards nailed horizontally to the braces *i*, two-thirds up their length, and vertically to the cap to cover the rest of the open space. A Henrich cooler is doing satisfactory work at the Deadwood and Delaware Smelter, Deadwood, S. Dak.

Blast.—The air blown into the furnace is not superheated. The use of hot blast has been repeatedly suggested of late and careful experiments in this line may lead to useful results here, although the experiments made in Germany* many years ago showed no saving. The machines that furnish the blast belong to the class of rotary positive-pressure blowers. Fans are not used, as they do not furnish sufficient pressure. Cylinder blowers may replace the rotary-pressure blowers, as they are not more expensive† than the pressure blowers, if the back leak or slip of the latter with a blast pressure of from 1 to 2 lb. be taken into consideration. The pressure blowers at present in general use are the Baker, the Root and the Cycloid. The Baker blower was at one time almost exclusively used, but the Root and the Cycloid have replaced it to a considerable extent, and the Cycloid is becoming a strong rival of the Root.

A furnace of from 36 by 60 to 36 by 84 in. will require a No. 6 blower; a furnace varying from 36 by 100 to 36 by 120 in. a No. 7; and with a distance of 42 in. between the tuyeres a No. 8 is required. These sizes give more blast than is actually necessary, but it is advisable to have a slight excess. With tight joints the blower may be one number lower.

There are two ways to supply the blast to a number of furnaces. Each furnace can have its own blower or several blowers deliver the compressed air into a blast-main, from which the single furnaces are supplied. The first method is the more desirable, as a furnace can be run independent of its neighbors, while at the same time it can be cross-connected with a main leading to the other furnaces and assist them in time of need. The second

*Merbach, "Die Anwendung der erwärmten Gebläseluft im Gebiete der Metallurgie." Leipsic, 1840, pp. 157-166, 168-170, 178-180; Le Chatelier, *Annales des Mines*, 1840, xvii., p. 3; Wedding, *Zeitschrift für Berg- Hütten- und Salinen-Wesen in Preussen*, 1871, xix., p. 155.

†Vezin, "Transactions of American Institute of Mining Engineers," xxvi., p. 1095.

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method is the one generally accepted, as the plant is cheaper, the care of it easier, and the repair smaller. The pressure in the single furnaces with the second method is regulated by a gate in the branch pipe. All blast pipes are made of galvanized iron. A common size is No. 18 iron; they are riveted with rivets set 1½ in. apart and soldered to make them strong and air tight. The main blast pipe or induction pipe *s* (Fig. 156), which receives the wind from several blowers, has a diameter that is from one-sixth to one-third larger than the combined outlets of the blowers. It usually runs along back of the furnaces near the dust chamber, and is suspended 8 ft. or more above the furnace floor in a wooden frame. It has safety-valves, and is closed at both ends by blast-gates. If one of the furnaces is to stop for a few minutes, one of the gates is slightly opened and some of the blast allowed to go to waste instead of slackening the speed of the engine, which it is generally inconvenient to do. In each branch pipe connecting a blower with the main pipe there ought to be a gate, to be closed when the blower is not in use, as otherwise it may happen that the idle blower will be running backward. From the main pipe branch pipes, each with its own gate, furnish the blast to the bustle pipes of the single furnaces (Figs. 160, 161, 166, 172, 174, 176). Back of the gate each branch pipe has an opening, with a thimble, to be connected with the pressure gauge by means of a ½-in. gas pipe brought down upon an adjoining post.

The pressure of the blast is measured by quicksilver, water or glycerine gauges, a difference of level of 2 in. of quicksilver, 28 in. of water, and 22 in. of glycerine (sp. gr. 1.28) equaling 1 lb., or 16 oz. avoirdupois, pressure per square inch. Water and glycerine are often colored or a drop of oil is added to each limb that the level may be plainly seen. Most works to-day have in the blower-room a recording gauge which gives the variations of the pressure of the blast in 24 hours. The gauge of the Bristol Co., Waterbury, Conn.,* is the one most frequently found.

From the bustle pipe thimbles pass downward to be connected by the wind-bags *q* with the tuyere pipes *p*. The connection is made by wiring. The thimbles usually have a slide-damper to permit the shutting off, wholly or partly, of the blast from the tuyere. The wind-bags are of closely-woven canvas that has

* *Engineering and Mining Journal*, Jan. 7, 1893.

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been soaked in water-glass, alum or mineral paint to prevent it from readily catching fire from a spark. With high pressure blast the leakage through the canvas has become great; it can be stopped by coating the inside with a thin cover of glycerine glue; at many works, however, the canvas bag has been replaced by a pipe of galvanized iron having a wind-gate to shut off the blast.

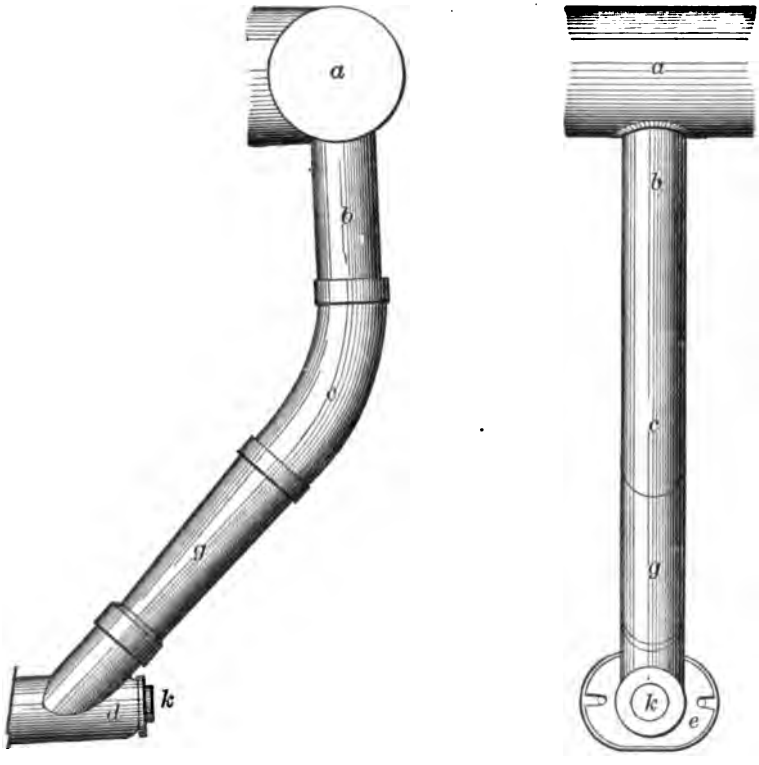
The number, size, and form of tuyeres and tuyere pipes vary a great deal. Hahn* reckons one tuyere with a 3-in. opening as being sufficient for 2 sq. ft. hearth area. Thus a 3 by 5-ft. furnace would require seven tuyeres; one that is 3 by 6½ ft., nine tuyeres. They are so distributed that one tuyere enters at the back and three or four are inserted symmetrically on each side. At some works the diameter of the tuyeres has been reduced to 2½ in.; at others it has been increased to 3½ in., and even 4 in. The tuyere at the back has been abolished at most works, while the tuyere at the front (Figs. 160 and 161) was discarded many years since as being very much in the way and of no special advantage. Blowing only from the sides has the effect of making the breast hard; blowing from the sides and ends of chilling in the center. The former difficulty can be avoided by having the last tuyeres close to the ends, and also by the manner of feeding; the latter is not easily remedied. Hence it is advisable to leave out the end tuyeres.

The ordinary blast pipe is made of galvanized iron. The horizontal arm, varying from 2 to 14 in. in length (Figs. 160 and 166), is either slightly conical and fits into a brass nozzle p' (Fig. 174) inserted into the tuyere-hole, or it is cylindrical and is soldered to the nozzle. The elbow which joins the other end to the wind-bag has a brass nipple soldered to it, which forms the eye or peephole s' (Figs. 166, 172, 174). This is closed either by a slide or by a cap having a glass or mica plate in the center, or simply by a wooden plug. In the center of the plug is left a small opening, the size of a pencil, to be closed by a piece of wood. This must be removed to observe the condition of the tuyere. To keep the blast pipe in its normal position and to thus prevent it from delivering the blast upward, which is its natural tendency, an iron band, hooked by means of two springs to the jacket, is passed around the elbow, or an iron loop is

* "Mineral Resources of the United States," 1882, p. 336.

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soldered to its inner side, by means of which it is hooked to the jacket. In order to close the tuyere-hole, when the pipe has been temporarily removed, a tuyere-cup has in many works replaced the ball of clay commonly used. It is made of galvanized iron, has the form of a nozzle, and is closed at the back, to which is soldered a handle.



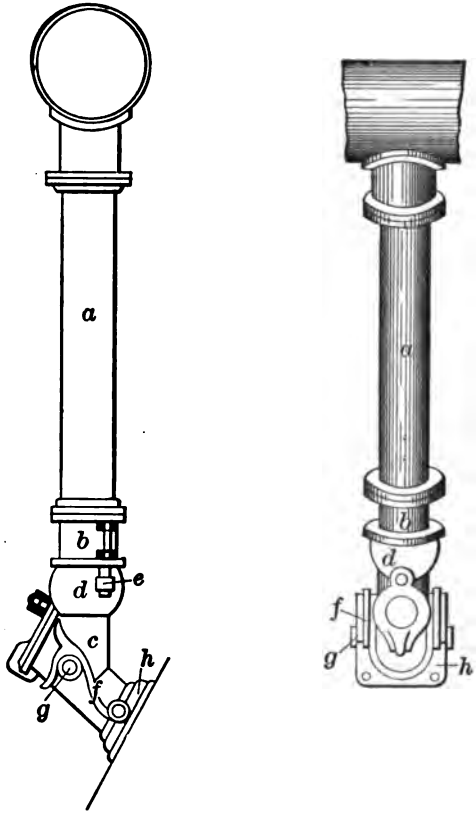
FIGS. 208 AND 209.—THE AUSTIN TUYERE.

In this connection may be mentioned Werner's adjustable tuyere pipe.* It consists of a cast-iron pipe to which a cast-iron elbow is fastened on the upper side by means of a hinge. The pipe is hooked with three springs to the jacket, and, working in a ball-and-socket joint, can be turned in any direction,

* Emmons, "Geology and Mining Industry of Leadville," monograph xii., U. S. Geological Survey, p. 682.

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and thus deliver the blast wherever desired, the springs keeping the pipe in position and very little air being lost by leakage. Other advantages are that should any slag run into the pipe it will not damage it and can be easily removed by raising the elbow. If the furnace is to be shut down for a short time, which usually necessitates the removal of the pipes, a piece of thin



FIGS. 210 AND 211.—THE UNZIGER TUYERE.

cardboard inserted between the flanges at the junction of pipe and elbow is sufficient. The writer has used this tuyere pipe, and while the supposed advantage of being able to set the blast in any direction is imaginary, because the air under pressure does not go as the tuyere is pointed, but in the direction of least resistance, its other features make it a good apparatus, preferable to the ordinary pipe of galvanized iron.

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A very simple and satisfactory tuyere pipe is that of Austin, shown in Figs. 208 and 209.* From the bustle pipe *a*, the thimble *b* passes downward in the usual way, having at its lower end the canvas tuyere-bag *c*. To this is fastened the sleeve *q* of light cast iron, turned conically on the inside at the lower end that it may make a tapering joint with the elbow of the cast-iron tuyere pipe *d*, similarly turned. It is thus out of the reach of any molten slag and is not in danger of being marred when a bar is driven into the tuyere pipe. The pipe *h* is bolted to the jacket with its planed colar *e* and two studs *f*. It has on its lower sides a slag-escape *i* (not shown) 1½ in. in diameter, closed by a wooden plug or a ball of waste. The peephole *k* is closed by two bushings and a plug, the latter having its $\frac{1}{16}$-in. peephole closed by a glass plate. By removing the plug and the bushings, openings 1¼, 2¼, and 3 in. in diameter are obtained. By loosening the nuts *f* the tuyere pipe is easily removed to clean the opening in the jacket.

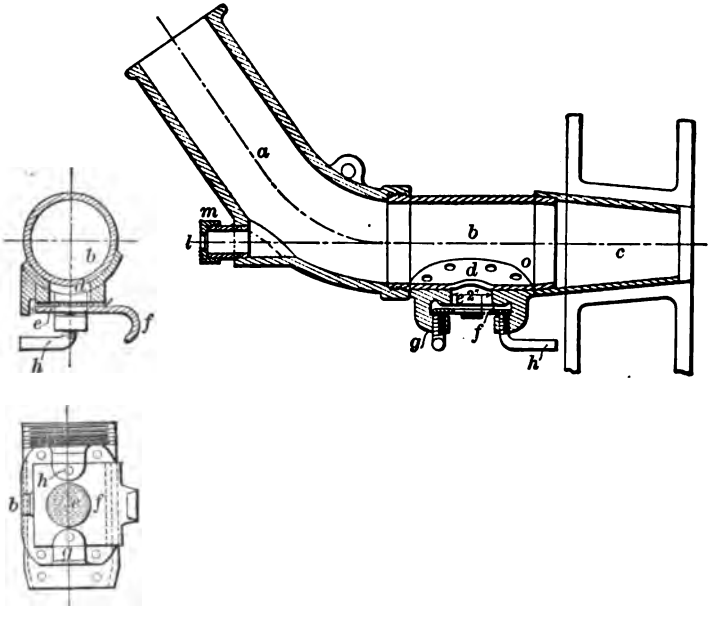
Another form of tuyere, with galvanized-iron pipe, is the one devised by Unziger (Figs. 210 and 211). It is used more in matting than in lead furnaces. To the lower end of the sheet-iron pipe *a* is fastened a cast-iron casing *b*, having the wind-gate. From the casing a cast-iron elbow *c*, with ball-joint *d*, is suspended by two spring-bolts *e*, revolving around trunnions *f* placed diagonally that the elbow may be swung sideways; the bolts press the ball of the elbows against the socket of the casing and keep the joint air-tight. The elbow is hooked by means of tightening hooks *f*, passing over trunnions *g*, to the collar *h*, bolted to the jackets.

In furnacing, slag will sometimes run into the tuyeres, and if it is discovered at the time, the accident can be remedied without difficulty. The Davis slag-escape† (Figs 212–214) notifies the furnace-man of the trouble. It consists in an opening in the underside of a tuyere pipe closed by a linen or paper diaphragm. As soon as any slag entering the tuyere touches the diaphragm, it burns a hole in it, and the noise of the escaping blast calls the attention of the keeper to the accident. Referring to the figures, the cast-iron elbow *a*, with peephole *l*, closed by cap *m*, is screwed to a piece of 4-in. wrought-iron pipe which has the opening *d* for the slag-escape and is riveted to the conical brass nozzle

* Taken from a drawing of the E. P. Allis Co., Milwaukee, Wis.
† *Engineering and Mining Journal*, Oct. 14, 1893.

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c, entering the furnace. To the underside of *b* is riveted the flange with its lugs *g*. These receive in a suitable recess the perforated piece of sheet iron *f*, having at its upper surface a bit of linen or paper *e*, and the screws *h* press it against the seat, thus making an air-tight joint. If slag entering the tuyere pipe burns a hole in the diaphragm, this can be replaced in a moment by loosening the screws *h* and drawing out the sheet iron. In the ordinary blast pipe, even with the best care, a leakage of air cannot be prevented. This has led some works to

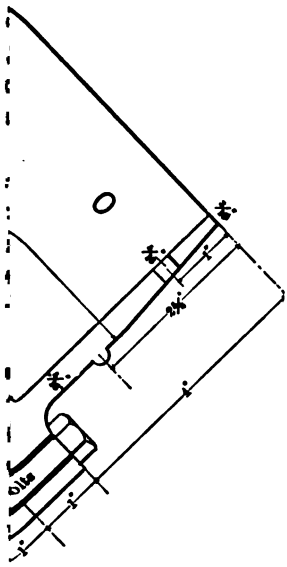


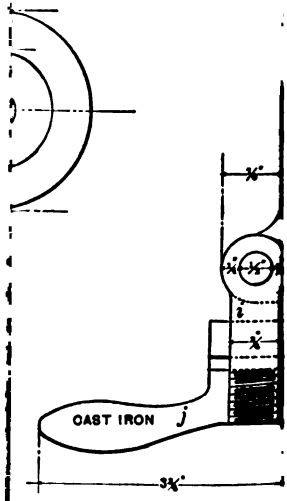
FIGS. 212 TO 214.—THE DAVIS SLAG-ESCAPE ATTACHED TO THE ORDINARY TUYERE.

adopt a cast-iron elbow or tuyere-box, which is fastened by an air-tight joint to the jackets. The constructions of Austin and Unziger form the transitions between the ordinary tuyere pipe and the tuyere-box. The constructions of Eilers, Murray, and Devereux may serve as examples of the latter.

Figs. 193 to 196 represent the automatic tuyere valve designed by Eilers. It consists of a cast-iron box *a''*, which is fastened with cap-screws to the jackets, as shown in Figs. 175, 176, 188-191. On the upper side is a cast-iron nipple *p*, to which the

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TUYERE BOX.

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wind-bag is fastened; on the lower side is a 3-in. opening through which chilled slag can be removed, to be closed with a plug. At the rear end are two openings, *l''*, the peephole and poking-hole, and *m''*, the outlet for the back pressure of the furnace gases when the blast is shut off. When the furnace is running, the valve *n''*, swinging on the cross-pin *o''*, and balanced outside of the box by the movable weight *p''*, is pressed by the blast entering through *p* against *m''* and closes this; as soon as the blast is taken off, the weight *p''* turns the valve, closes *p*, and opens *m''*, thus preventing the gases in the furnace from entering the blast pipe and furnishing them an outlet into the open air. However, if any slag accidentally gets into the valve, it is difficult to remove it, and the valve is liable to become warped.

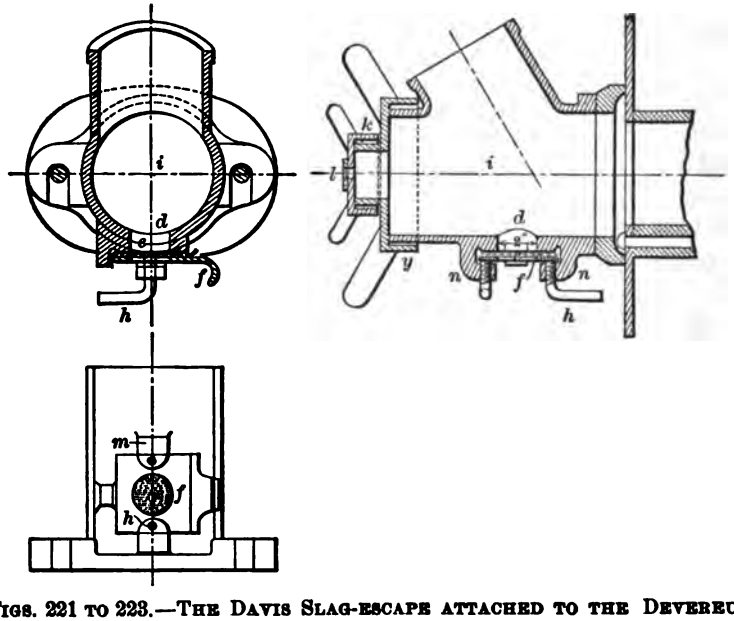
Murray's tuyere box is shown in Figs. 215 to 220. Fig. 218 shows an obtuse-angle elbow. The opening *a*, through which the blast enters the jacket, is rather large—5 in. in diameter. Opposite is the poking-hole *b*, closed by the cap *c*, which contains the eyehole. Between the tuyere pipe *d* and the cast-iron nipple *e*, over which passes the wind-bag, is the wind-gate *f*. Of special interest is the drop bottom *g* of the belly-pipe *h*. It is held in place by the hinged bolts *i* and the crank-nuts *j*. The drop bottom has a fusible plug *k*, a thin disk of lead, which will melt as soon as any slag enters the belly-pipe, and thus call immediate attention to the accident. The elbow is fastened to the jacket by slipping it over four threaded bolts *m* that have been screwed into the jacket, and then tightening it with nuts.

The characteristic of the Devereux tuyere box* is that the blast can be made to play in different directions. The tuyere hole is lined with a cylindrical bronze tube in which can be revolved an iron sleeve having a diagonal bore. By turning this sleeve the blast can be directed up and down and to right and left; it cannot, however, be directed centrally and so the tube has fallen into disuse. A Devereux tuyere box with Davis slag-escape is shown in Figs. 221–223.† The tuyere box *i* is closed at the back by the cover *y*, the collar of which has a cap *k*, with peephole *l* in the center. The slag-escape differs from that shown in Figs. 212 to 214 in having the lugs *n* cast on the wind-box and not riveted to it, thus greatly simplifying and cheapening the arrangement.

* Patent No. 318,604, May 26, 1885. † *Engineering and Mining Journal*, Oct. 4, 1893.

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Slag-Pots.—The disposal of slag, which with small furnaces and charges containing little matte was a comparatively simple matter, has become more and more complicated. Twelve ordinary slag-pots, 24 in. in diameter and 15 in. deep (1.6 cu. ft. capacity), were sufficient for a 33 by 100 in. furnace. Now, with a 42 by 120-in. furnace and an increased matte-fall, if the slag were disposed of in the old way or by pouring, the number of pots would have to be much increased or their size enlarged. The last has been done at one place where their capacity has



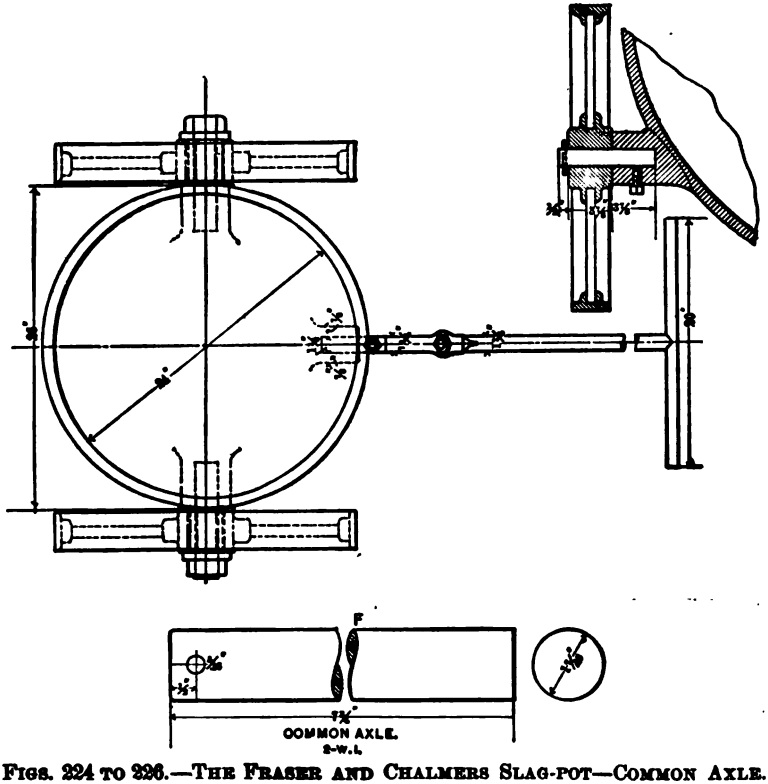
FIGS. 221 TO 223.—THE DAVIS SLAG-ESCAPE ATTACHED TO THE DEVEREUX TUYERE BOX.

been doubled. This has been made possible by covering not only the front of the furnace floor, but the entire dump to where the slag is to be poured, with cast-iron plates, 36 by 40 by 1 in. The effort required to pull the large pots over the smooth surface is not greater than for small ones over the dump approximately smoothed by slag-squares. The necessity for handling large amounts of slag and settling out completely the increased quantities of matte, especially difficult with zincky ores, has given rise to improvements in the old methods and the introduction of new ones.

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Figs. 224 to 226 show the usual paraboloid form of a slag-pot with the ordinary compression-spoke wheel, the spokes being wrought iron, the axle machine steel. The length of the handle is 5 ft., the height of the cross-piece above the ground 2 ft. 8 in.

The form of the bowl varies somewhat with the use the pot is put to; if it is to be used for dumping, the sides will be slightly rounded, if for settling out matte, the sides will be straight. Pots are sometimes made to-day of what is commercially called



FIGS. 224 TO 226.—THE FRASER AND CHALMERS SLAG-POT—COMMON AXLE.

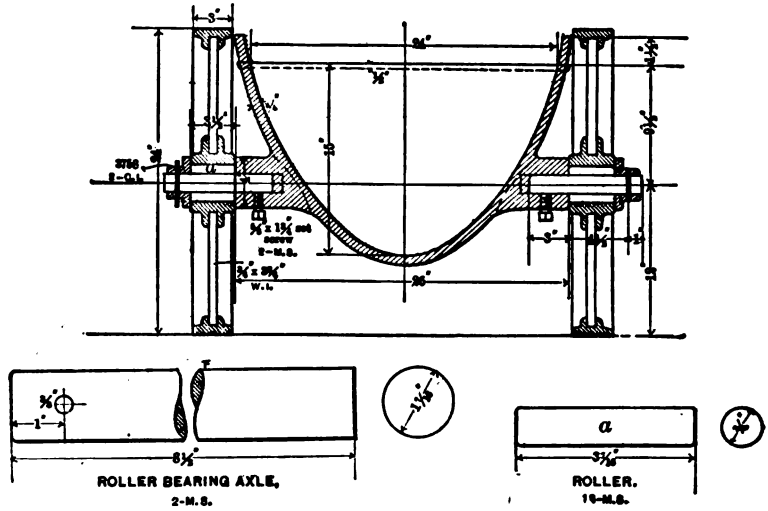
"semi-steel." This is a mixture of two-thirds No. 1 charcoal iron and one-third of varying proportions of good wrought-iron scrap, soft-steel punchings, and shearings and rail butts.

In order to lessen the friction at the hub, roller bearings, *a* (Figs. 227 to 229),* have sometimes been introduced with satis-

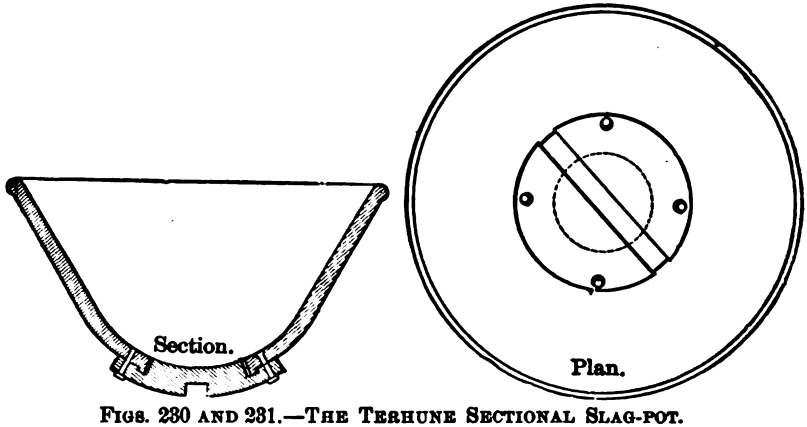
* Taken from a drawing of Messrs. Fraser & Chalmers, Chicago, Ill.

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factory results. In other cases they have not proved worth the additional expense. The axles are as a rule keyed into their sockets.



FIGS. 227 TO 229.—THE FRASER AND CHALMERS SLAG-POT WITH ROLLER BEARINGS.



FIGS. 230 AND 231.—THE TERHUNE SECTIONAL SLAG-POT.

Terhune* (Figs. 230 and 231) has made the bottoms of slag-pots movable, so that they can be replaced when corroded or cracked. Matte often sticks to their joints, however, and is hard to remove.

* "Transactions of American Institute of Mining Engineers," XV., p. 92.

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An ordinary slag-pot lasts about one year.

The Iles-Keiper pot* is a large overflow-pot which retains the matte, while the slag runs over through a spout in the side into an ordinary slag-pot. To prevent the melted mass from solidifying the pot has a cast-iron cover. Overflow-pots,† like those

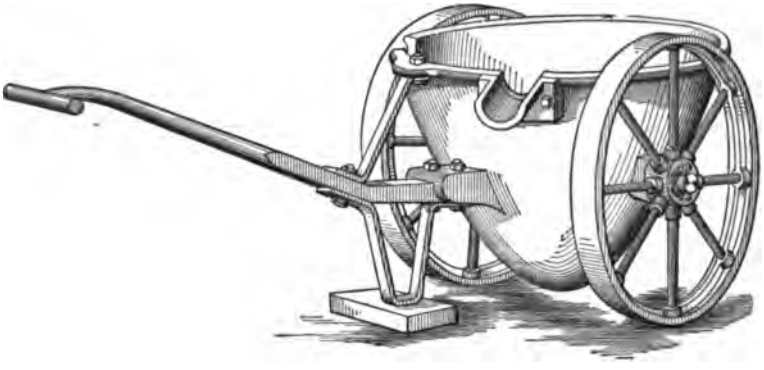


FIG. 232.—OVERFLOW SLAG-POT.

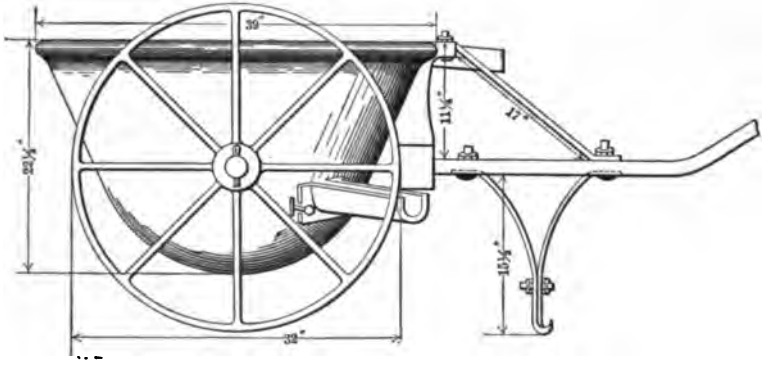


FIG. 233.—OVERFLOW SLAG-POT WITH MATTE-TAP.

in Fig. 232, have been used for a good many years. At some works the slag is allowed to harden in them from the surface down to a depth of several inches, the inlet and overflow for the slag being kept open. Thus the hardened slag, taking the place of the iron cover, prevents the liquid slag below from cooling, and promotes a good separation of matte and slag.

* Patent No. 335,224, Feb. 2, 1886.

† Catalogue of the E. P. Allis Co., Milwaukee, Wis., "Mining Machinery," p. 71.

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When such a pot has been filled with matte it is wheeled to the dump and allowed to cool. Sometimes there is a tap-hole near the bottom to remove, after the matte has solidified, any lead that may have run out with the melted masses. To avoid the necessity of moving the heavy overflow-pot when filled with matte, there is often a tap-hole (Fig. 233)* through which the matte is periodically discharged into an ordinary slag-pot. As this tap is liable to wear away, Terhune (Fig. 234) has made it detachable, the casting, with tap-hole and spout, being replaced in a few minutes. Movable, brick-lined forehearths (Figs. 235–236) come

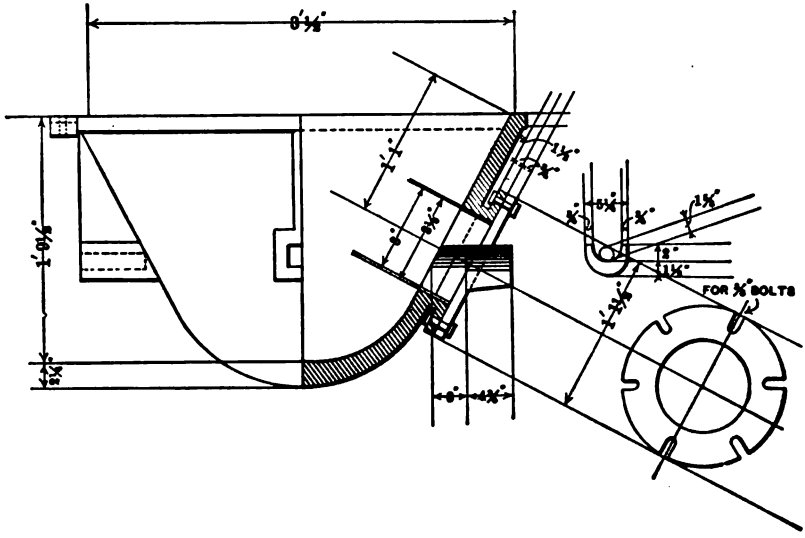


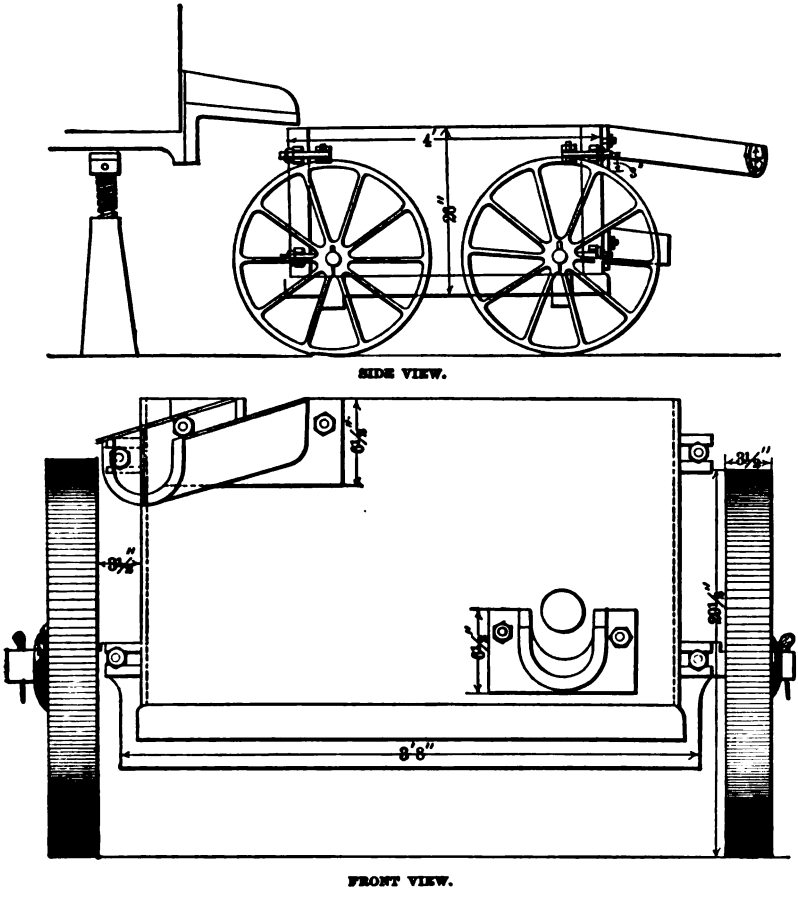
FIG. 234.—OVERFLOW SLAG-POT WITH TERHUNE'S DETACHABLE TAP.

more into play in concentrating matte than in ore smelting, as it is necessary to produce large amounts in order to prevent it from cooling on the bottom and building. This is especially the case with high-grade matte. In tapping, it may be collected in a flat receiver, as it is then easily broken up into sizes that can be conveniently handled. Such a "shell" for matte, designed by Rhodes, is shown in Figs. 237 and 238. Low-grade matte is collected in the ordinary slag-pot, as it quickly disintegrates when exposed to the air and is easily broken. The Orford siphon-tap,† from

* Braden, "Transactions of American Institute of Mining Engineers," xxvi., p. 38.
† Peters, "Modern Copper Smelting," New York, 1895, p. 294.

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which matte and slag are discharged in a continuous stream, is not, so far as the writer is aware, used in connection with lead-smelting. At the Arkansas Valley Works, Leadville, Colo.,* there is in use a large reverberatory smelting furnace, designed by Rhodes,



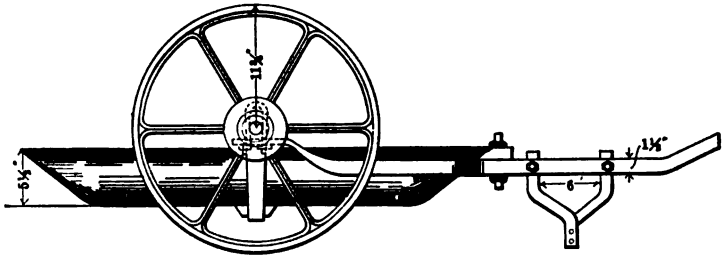
FIGS. 235 AND 236.—MOVABLE BRICK-LINED FOREHEARTH.

with water-cooled sides and air-cooled bottom and holding as much as 100 tons of slag and matte. The slag-matte mixture, as collected in the ordinary slag-pot from the blast furnace, is poured into it near the flue and given time to settle. By keeping fuel

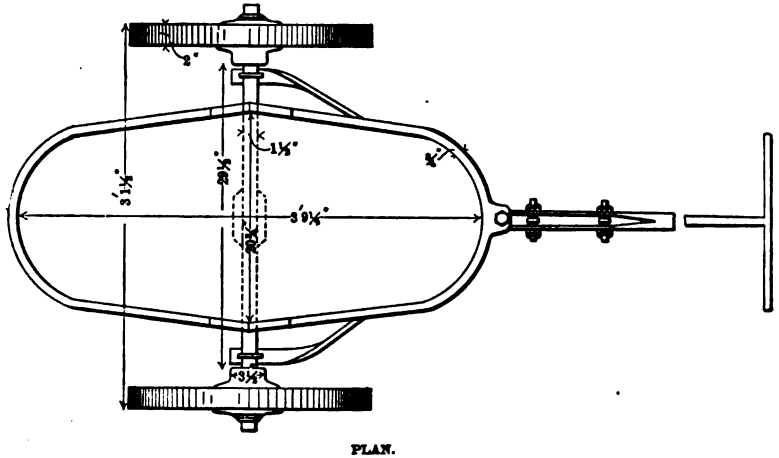
* Kloz, *Engineering and Mining Journal*, April 10, 1897.

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on top of the liquid slag, the ferrous oxide is prevented from becoming oxidized more highly and any magnetic oxide present is reduced to the ferrous state, thus facilitating the separation. On the bottom of the hearth is placed some scrap iron, the presence of which prevents the lead from becoming coppery and making the tapping difficult. Matte and slag are tapped near the fire-bridge from different levels. Little fuel is required to keep the



SIDE VIEW.



PLAN.

FIGS. 237 AND 238.—THE RHODES SHELL FOR MATTE.

slag fluid. With 400-oz. bullion, slags containing 10 oz. silver per ton and 6% lead have been cleaned so as to assay less than 1 oz. silver and 1% lead, and the waste slag from the furnace never runs higher.

A third method, and the one in common use for preventing a loss of metal by shots of matte adhering to slag, is to allow the matte to settle in a catch-pot and then to tap the still liquid slag

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above the level of the matte. Thus the matte in the bottom of the pots remains undisturbed, and the shell of chilled slag (that inclosed the liquid slag) is recovered and smelted over again, as it is the only part of the entire slag that is liable to be rich. A pot of this description generally goes by the name of Devereux pot, Mr. Devereux having taken out several patents* for different applications of this method.

The principle involved is, however, of earlier origin than is generally supposed. Kerl† states that such a pot was already in use in 1873 at Schemnitz, Hungary; Painter‡ reports its use at Stolberg, Prussia, at the same date, and Vivian§ as early as 1846 tapped the liquid slag from cones placed on the dump and saved the sculls, which alone were resmelted. A Devereux pot, designed by Keller, and somewhat modified by Murray, is shown in Figs. 239 to 244, and will serve to illustrate the third method. The pot has a cast-iron bowl of the usual paraboloid form, but 3½ in. above the bottom is the tap-hole *a*, through which the liquid slag is discharged after the matte has settled out. To prevent injury to the pot at the tap-hole, when this is opened with a steel bar, the casting is made thicker by a ring *b*, ending in a rib *c* which reaches the top.

In this pot is seen a third class of wheel. The hub *d* has alternate spoke-sockets *e* and *e'*, similar to those of the tension-spoke wheel. This wheel is more common than the ordinary compression-spoke wheel, as it is stronger.

Cole, Gaylord and Keller have patented‖ this combination of cast-iron spokes with a wrought-iron tire. The spokes are threaded at one end and screwed into the cast-iron hub, where they are held in place by jamb-nuts. The opposite ends have flanges, on to the outer sides of which a strong wrought-iron tire is shrunk and then fastened by countersunk set-screws. The work of the Devereux pot is shown by the following examples: The run-slag assayed in one instance 3 oz. silver per ton, the shell 10 oz., the waste slag 1 oz.; in another case the shell sample showed 24 oz. silver per ton and 5% lead, while the waste slag ran 2 oz.

* U. S. Patent No. 312,439, Feb. 17, 1885; No. 335,114, Feb. 2, 1886.

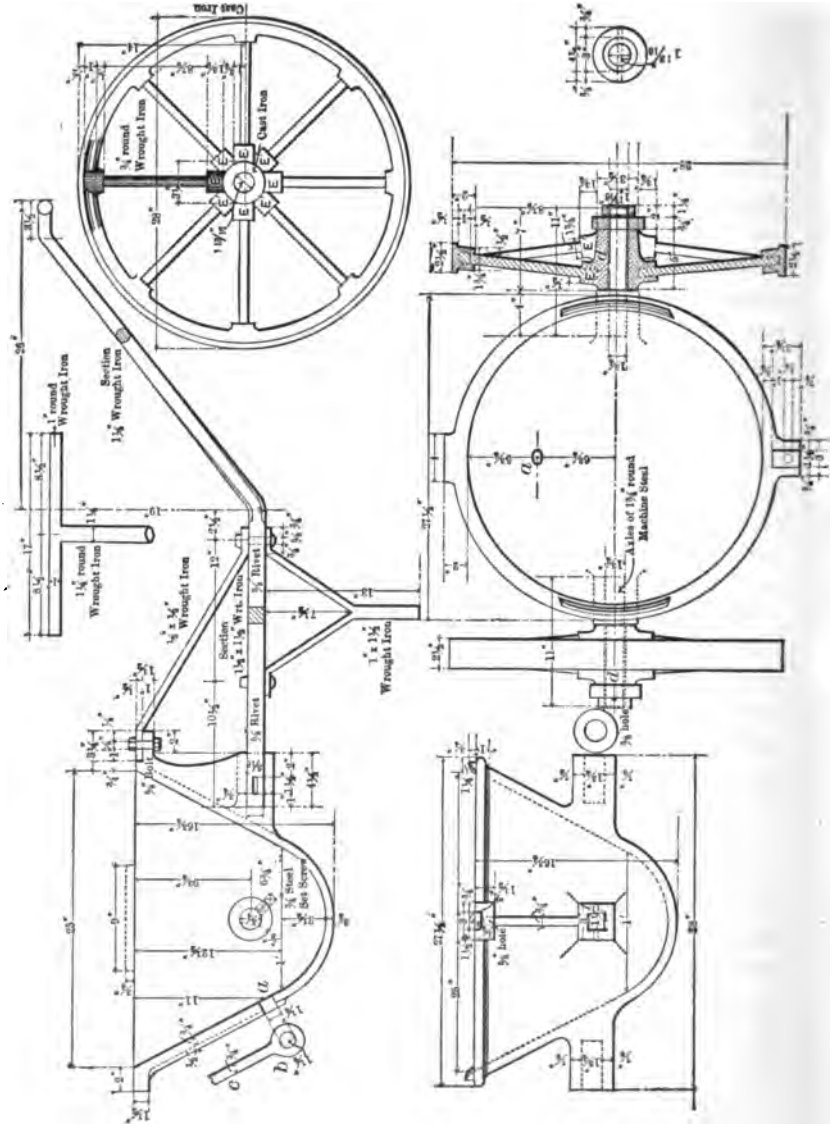
† "Grundriss der Metallhüttenkunde," Leipsic, 1881, p. 100.

‡ "Report on the Metallurgy of Lead, Silver, Copper, and Zinc at the Vienna International Exhibition of 1873," Washington, 1875, p. 61.

§ *Engineering and Mining Journal*, April 9, 1881.

‖ U. S. Patent No. 470,340, Nov. 29, 1892; "Transactions of American Institute of Mining Engineers," xxii., p. 578, ill.

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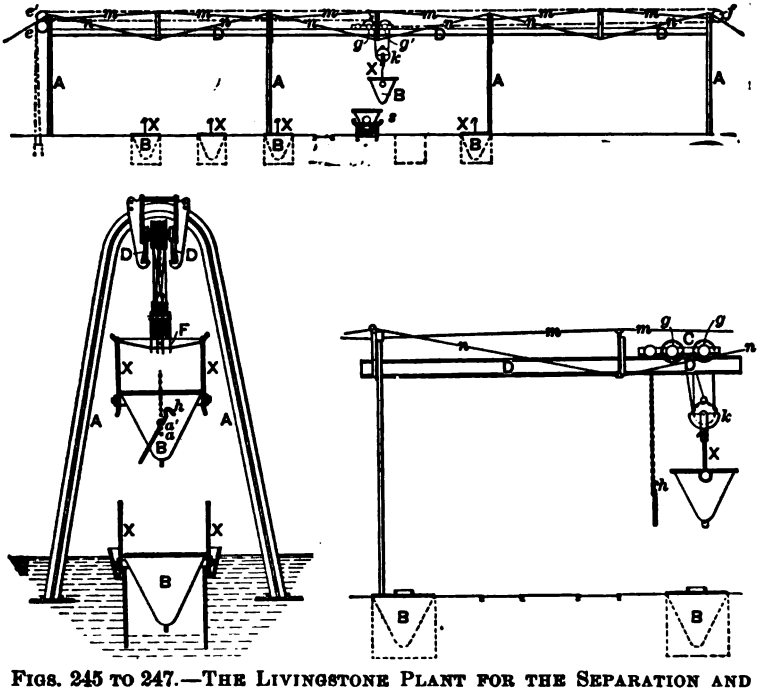


FIGS. 230 TO 244.—THE DEVEREUX SLAG-POT.

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silver and 1% lead. The values of the shell are unevenly distributed as shown by the following figures: top, 3 oz. silver; side, 2 oz.; bottom, 3.4 oz.; the waste slag assaying 1.2 oz. silver per ton and 0.4% lead.

Since smelters have been forced to treat charges rich in zinc, it has been found that the small-sized pot does not always separate matte and slag satisfactorily. Livingstone designed and



FIGS. 245 TO 247.—THE LIVINGSTONE PLANT FOR THE SEPARATION AND DISPOSAL OF SLAG.

introduced at the works of the Omaha and Grant Smelting and Refining Co., Denver, Colo.,* a system of large settling pots which effectively separate all the matte and slag from a plant having a daily capacity of 350 tons of ore. A sketch of the plant, a side elevation showing the general arrangement, is given in Fig. 245, an end elevation in Fig. 246, and a side elevation of the overhead traversing crane in Fig. 247. On the dump and parallel with the row of furnaces are five pits, about 5

* Austin, *Engineering and Mining Journal*, Nov. 23, 1895.

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ft. in diameter and 5 ft. deep, which receive the five separating pots *B*, having a capacity of about 4 tons of matte and slag. The pits are beneath the overhead track *D* (15-in. I-beams), which is supported by the frame *A* made of 60-lb. rails. To the trunnions of a separating-pot *B* are attached hooked links *x*, by means of which it is suspended from the cross-bar *F*, which is raised and lowered by the running block *K* and the traveler *C*, the ropes running over sheaves *g g'* on traveler to end-sheaves *e e'* and *f*, *e'* serving to raise and lower the pot, *e'* and *f* to change the position of the traveler; *m* and *n* are tie-rods for stiffening the track, *S* is the waste-slag car. The mode of operation is as follows: Each slag-man from a blast furnace wheels his ordinary slag-pot to a separating-pot and empties it, reserving for re-smelting any shell that has formed. When a settling-pot has been filled the matte is allowed to settle for from 10 to 15 minutes, the pot is then raised and run to where stand the waste-slag pots *S*. The slag is now tapped through one of the two tap-holes *a a'* in the side and run into the waste-slag pot to be transferred to the edge of the dump and poured. The settling-pot, which still holds all the matte with some slag covering it, is taken to a large pot, holding about 1,000 pounds of matte, and tapped into it from the bottom. When cool, the cake of matte is rolled out and the slag separated and reserved for re-smelting. The empty settling-pot is brought near to the end of the overhead track and raised until the hook *h*, suspended from the carriage, can be inserted in the staple at the bottom and tilted, when the shell of rich slag will slip out on to the ground to be re-smelted. It is then raised, unhooked and returned to its pit ready to be filled again. For operating the plant are required on the day and night shift, one man at the engine, one tapper, one driver and mule, and a man at the end of the dump, all working 12 hours, and in addition eight 10-hour men on the day shift alone who handle the sculls and matte made in 24 hours. The objection made to the Livingstone method is that a blast furnace works irregularly, making sometimes much matte and again very little, while even with the two tap-holes there is not chance enough for free play in changing the level at which the slag ought to be tapped, with the result that either matte is sometimes run off with the slag or too much slag has to be re-smelted. The objection sounds well theoretically, but as a matter of fact seems to be out of place.

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In disposing of the large amounts of slag made to-day two methods are followed. One is to have large receiving pots, which are poured; the other is to granulate the slag. Occasionally cones of slag are allowed to cool on the dump, broken up by sledges and crushed to a size suitable to be used as railroad ballast. Granulated slag frequently serves for the same purpose. Two large receiving pots made by the Colorado Iron Works, Denver, may serve as examples. The Nesmith Single-Bowl Dumping Car is represented in Fig. 248. It is 13 ft. 6 in. long, 6 ft. wide and 4 ft. 7 in. high from top of rail to top of bowl, and weighs 6,300 lb. The bowl, having a capacity of 35.16 cu. ft. and holding about 3 tons of slag, is suspended from trunnion-axles to which are keyed gear-wheels *y*, meshing into racks on the supporting frame *o o'* of the car, and a screw gear *z*, operated by a screw *w*, turned by a hand crank. The bowl is rolled over on its

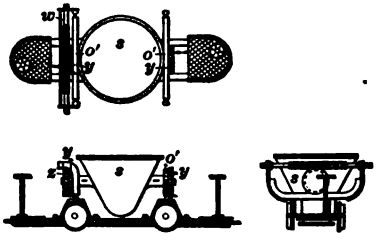


FIG. 248.—THE NESMITH SINGLE-BOWL DUMPING CAR.

wheels far enough to pour the slag over and beyond the track, and to allow the shell of slag to follow it. The same firm has put on the market a slag-car with a single elliptical bowl whose center of gravity is such that when empty the bowl will stand right side up and when filled with slag will roll over, after the brake has been released. Henrich* recently published a drawing and description of a large slag-car of 25 cu. ft. capacity designed by him.

The Nesmith Double-Bowl Dumping Car† is shown in Figs. 249 to 254.

The car has two large tilting pots *A A*, each having a capacity of 7.38 cu. ft., and holds about 1,280 lb. of slag. The car consists of a truck-frame *a*, with platform *b*, brake *c*, and railing *d*, by which the driver holds on. The frame carries the boxes *e* for

* "Transactions of American Institute of Mining Engineers," XXV., p. 95.
† U. S. Patent No. 388,708, Aug. 28, 1888.

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the axles *f* of the wheels *g*. Two bridge-beams *h*, lying transversely across the frame *a*, serve as support for the frame *h'*, which carries the central pin-socket *m*, in which the pin *n* is made fast by the nut *n'*. The swinging frame consists of the channel-irons *i i* (held apart by the central blocks *j j*, in which are the swivel-eye and the end-blocks *j' j'*) and the beams *k k* (resting on the channel-irons); the latter have on their under side the bearings of the trunnions *l l* of the tilting-pots *A A*. These are pivoted out of center, so that when in their normal position they may lean against the stop-pieces *o*. They are held in position by the pawls *r*, with disengaging handles *r'*, which fasten into the teeth *q* of the projecting head or collar *p* of the trunnions. This collar also has the holes *s* for the operating-bar *t* (Fig. 254), with which the pots are tilted. The weight complete of truck and pots is 5,000 lb. When in their normal position the pots are placed as shown in Figs. 249 and 250. When they are to be emptied, the frame *iikk* is swung 90° on the swivel (Fig. 254), the pots are emptied together on both sides of the track; or, if the slag is to be discharged only on one side, the frame is returned to its normal place after one pot has been tilted and the other swung into position.

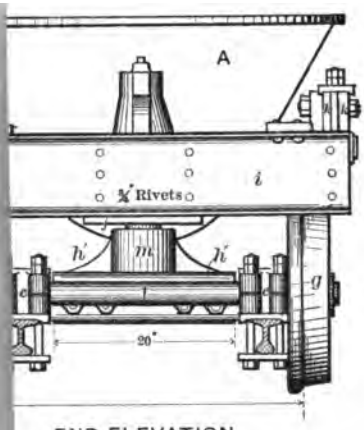
The granulating of slag is used principally at copper smelting works. It forms a very simple and cheap method of disposing of the slag and has in addition the advantage that it is easy to obtain a good average sample. Three successful granulating plants used at lead and copper-nickel works may serve as examples.

The granulating plant of the Hanauer Smelting Works,* Salt Lake City, Utah, designed by Terhune, is shown in Fig. 255. Its essential features are a wooden trough *a*, 2 ft. wide and 6 in. deep, inside measurements, the bottom of which is covered with ¾-in. cast-iron plates. A stream of water of considerable velocity running through it receives the slag, poured in from the top in a stream 1¼ in. in diameter, granulates it and carries it off into a water-tight pit *b*, from which a rubber bucket elevator *c* raises it 40 ft. into a storage bin large enough to hold the slag produced in 12 hours, say 100 tons, thus doing away with night-work in loading into the railroad cars, which carry away the slag to be used as ballast. The trough is placed underground at a pitch of ¼ in. to the foot and 10 ft. in front of the furnace build-

* *School of Mines Quarterly*, xv., p. 108; and private communications, 1896-97

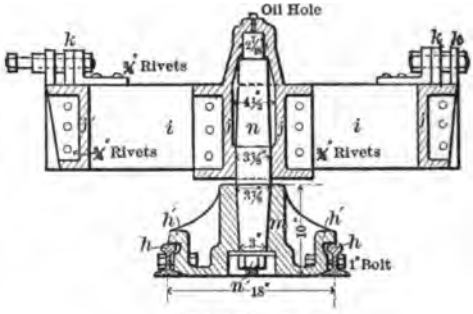
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FIG. 251.



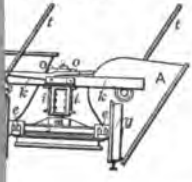
END ELEVATION.
FRONT POT REMOVED.

FIG. 252.



SECTION A B.
REVOLVED TO SIDE.

FIG. 254.



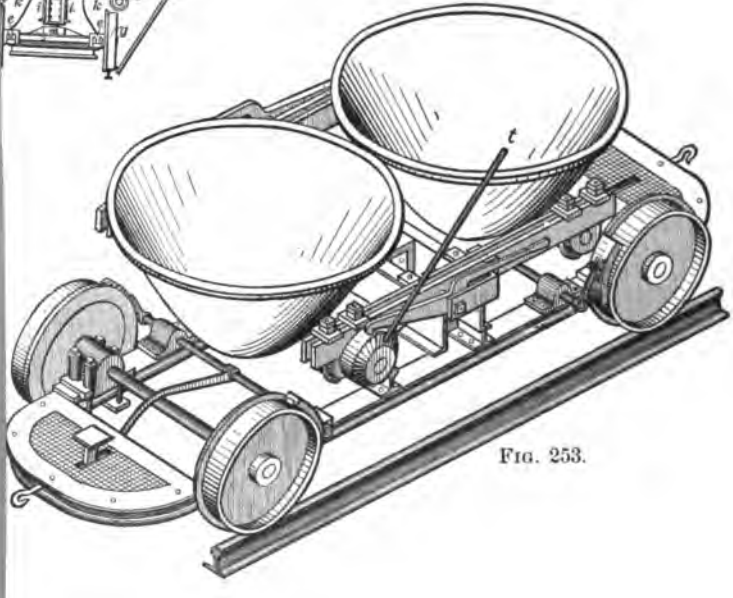


FIG. 253.

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ing and parallel with it. The depth of the chute underground where it receives the slag is somewhat less than 3 ft., as if deeper the slag solidifies on the bottom. Near the face of the dump are chutes, covered by flanged cast-iron plates, one to each furnace, for discharging the slag into the trough. The water, which arrives in the flume *d*, is discharged at the side through the closed horizontal box *e* into the inclined box *f*, which is open at the top. The amount of water required is 2,400 gal. per minute. Very little steam is generated and hardly any noise is made, provided no matte is carried off with the slag, as matte coming

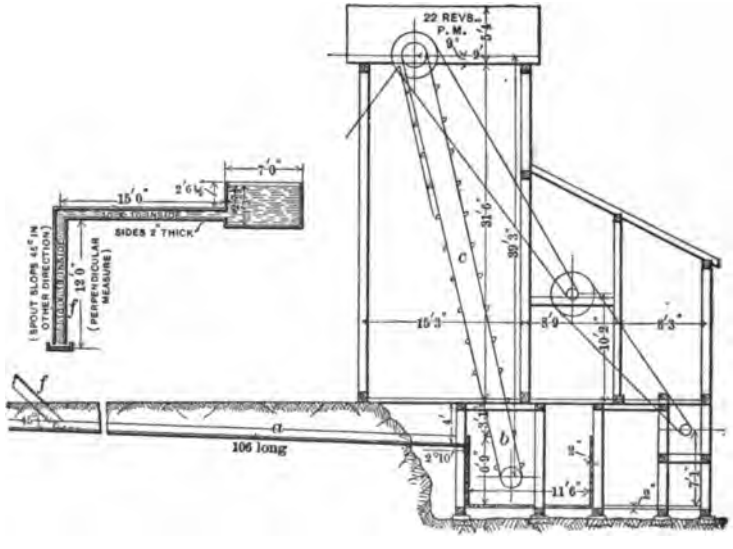


FIG. 255.—THE TERHUNE SLAG-GRANULATING PLANT.

in contact with water causes explosions. The cast-iron plates in the trough are worn thin in about six months. The elevator is driven by a 4-horse power Leffel mining wheel. The belt, traveling at a speed of 250 feet per minute, is 8 in. wide and 8-ply, and lasts about a year; the buckets are of malleable castings and have $\frac{3}{8}$-in. holes in the ends for the water to drain off. The slag granules are small enough to pass a $\frac{1}{4}$-in. sieve; whatever parts they strike will be quickly worn out, if not protected by iron plates.

At the Omaha works of the Omaha and Grant Smelting and

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Refining Co.* is a peculiar arrangement for chilling liquid slag that it may break into flat pieces ($\frac{1}{4}$ to 1 in. thick and $\frac{1}{2}$ to 4 sq. in. surface area), designed by Page, and in operation for a number of years. It consists of a pit about 35 ft. in diameter and 3 ft. deep filled with water, there being a constant in and out flow to keep the chilling water cool. Along the circumference of the pit and on the water travels an annular disk of cast-iron plates, about 2 ft. wide and $1\frac{3}{4}$ in. thick, and slightly trough-shaped on top, which is supported partly by radial arms from a central pillar and partly by rollers underneath which revolve on stationary plates supported by pillars. The water supply is so regulated that the bottom of the annular chilling-plate shall be in contact with the water, but that this shall not overflow into it. The apparatus makes one revolution in five minutes, the motion being given by a rack on the periphery of the disk and a pinion. The pit is inclosed by a ring of cast-iron plates fixed at an angle of about 60° from the horizontal and reaching over the revolving trough-shaped disk. On the pit-side the inclosing plates have a rim to stop the slag-pots. A pot filled with waste slag is wheeled on to the inclosing plate and the contents poured on to the revolving disk. The slag spreads out to the thickness of from $\frac{1}{4}$ to 1 in., cools quickly and breaks up into small fragments which are scraped off by a stationary piece of $\frac{1}{2}$-in. steel, set diagonally toward a conveyer discharging them into bins from which they are transferred to railroad cars.

The McArthur granulating trough of the Canadian Copper Co., Sudbury, Ontario, is shown in outline in Fig. 256 and in detail in Figs. 257 to 263.† The trough *a* (Fig. 256) is mounted on wheels so that it can be easily pushed to one side, when the movable forehearth *b* receiving the slag-matte mixture from the blast furnace *c* has to be exchanged, an operation taking about 10 minutes. The trough consists of two parts, the granulator proper *d* and the trough *e*, which conveys the granulated slag to the mouth of an underground flume (not shown) 50 ft. long which discharges it into a drainage-pit whence the slag is raised by means of a bucket elevator and emptied into railroad cars. The granulator *d* (Figs. 257 to 263) consists of a trough-shaped cast-

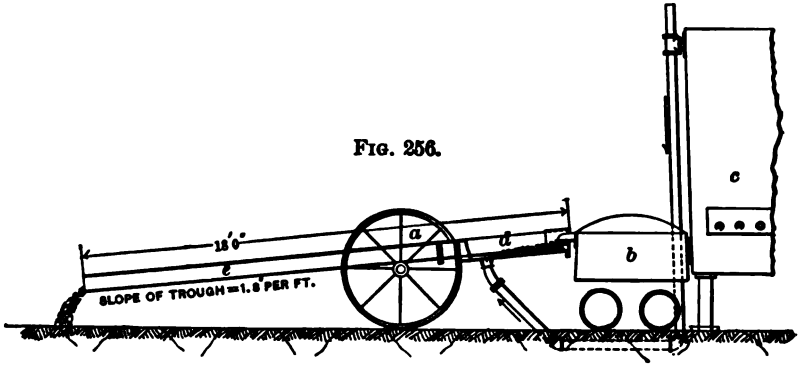
* Braden, "Transactions of American Institute of Mining Engineers," xxvi., p. 51; private notes, August, 1896.

† Drawn by W. Darlington from sketches of J. McArthur, January, 1897.

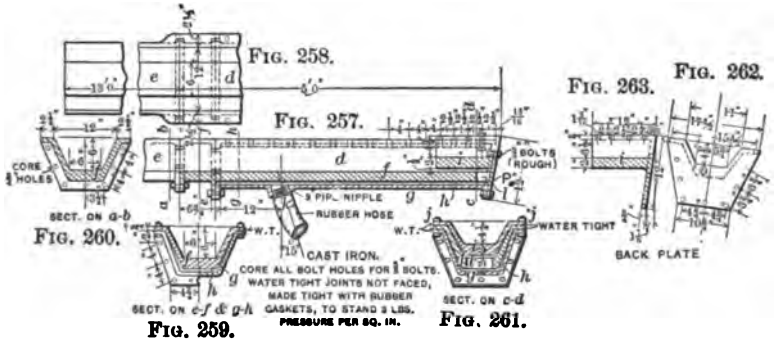
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ing *f* with side-flanges inclosed by a cast-iron jacket *g* of similar shape so as to leave a water-space *h* between them. To the upper end of the jacket is bolted the back plate (Fig. 257, 261, 263) with flanged spout *i*. Through the flanges pass bolts *j*, the joints being made water-tight with rubber gaskets. The granulating water, which is the blast-furnace water jacket overflow, and has a

FIG. 256.



SIDE VIEW.



DETAILS.

FIGS. 256 TO 263.—THE MACARTHUR GRANULATING TROUGH.

temperature of about 65° C., enters the water-space at *k*, rises through the opening *l*, and being deflected by the spout *i*, passes down under it in a thin stream. The slag flows in a similarly thin stream over the spout *i*, and slag and water come into quiet and smooth contact, both flowing in the same direction. There is no steam, splash or noise and no danger of explosion. As the slag is not dropped into the water, it does not adhere to the trough-bottom and thus form a semi-liquid ball sure to explode.

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The granulated slag is uniformly coarse. At present 100 tons of slag are granulated with 60 gal. of water per minute, having a head of 16 ft.; but the capacity of the granulator is half as great again as this. The apparatus is also suited for granulating matte, if this is tapped into a receiver and poured in a thin stream on the spout.

§ 68. LEAD SLAGS.*—As the object of smelting is to separate by fusion the lead as metal from its ore, the other constituents, the acid silica and the bases, iron, lime, etc., must be combined in certain proportions to form a slag. If this does not occur naturally, either silica or the various bases, *i.e.*, fluxes, will have to be added in the requisite quantities.

Composition of Lead Slags.—The main slag for the lead smelter is the singulo-silicate slag,

$$\overset{II}{2RO}+SiO_2, \text{ or } \overset{III}{2R_2O_3}+3SiO_2,$$

with the oxygen ratio of bases to silica as 1 : 1.

In practice the slags are made slightly more acid than the formula calls for. Another slag that is sometimes made is the sesqui-silicate,

$$\overset{II}{4RO}+3SiO_2, \text{ or } \overset{III}{4R_2O}+9SiO_2,$$

with the oxygen ratio of bases to silica as 2 : 3, written sometimes

$$(\overset{II}{2RO}+SiO_2)+2(\overset{II}{RO}+SiO_2), \text{or} (\overset{III}{2R_2O_3}+3SiO_2)+2(\overset{III}{R_2O_3}+3SiO_2) \text{ as}$$

if consisting of a singulo and a bi-silicate. This slag, not being so readily fusible as the singulo-silicate, is made, if a high temperature is desirable, *e.g.*, if much lead sulphide is to be decomposed by metallic iron resulting from ferric compounds. With a readily fusible slag the ferric compound would be reduced only to a ferrous compound and then slagged. Henrich† recommends a slag more acid than the singulo-silicate for arseni-

* Hahn, "Mineral Resources of the United States," 1882, p. 325; Guyard in Emmons' "Geology and Mining Industry of Leadville, Colo.," monograph xii., U. S. Geological Survey, p. 701; Iles, "Mineral Resources of the United States," 1883-84, p. 440; Schneider, "Transactions of American Institute of Mining Engineers," xi., p. 56; Keller, *Proceedings Institute of Civil Engineers*, London, 1893, cxii., p. 208; Struthers, *School of Mines Quarterly*, xvi., p. 356, partial bibliography.

† *Engineering and Mining Journal*, Oct. 6, 1883.

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cal ores in order to keep the speise liquid; *i.e.*, if its formation cannot be avoided.

The following tables by Balling* give the necessary proportions of silica and bases required to form singulo-silicate, bi-silicate and sesqui-silicate slag, one table having silica for a unit, the other the bases. The bases baryta and lead oxide have been added.

One Part by Weight of Silica Requires	Parts by Weight of Bases.	One Part by Weight of Bases Requires	Parts by Weight of Silica.
For Singulo-Silicates:		For Singulo-Silicates:	
Lime	1.86	Lime	0.535
Baryta	5.10	Baryta	0.196
Magnesia	1.33	Magnesia	0.750
Alumina	1.14	Alumina	0.878
Ferrous oxide	2.40	Ferrous oxide	0.416
Manganous oxide	2.36	Manganous oxide	0.422
Lead oxide	7.43	Lead oxide	0.134
For Bi-Silicates:		For Bi-Silicates:	
Lime	0.93	Lime	1.070
Baryta	2.55	Baryta	0.392
Magnesia	0.66	Magnesia	1.500
Alumina	0.57	Alumina	1.747
Ferrous oxide	1.20	Ferrous oxide	0.863
Manganous oxide	1.18	Manganous oxide	0.845
Lead oxide	3.71	Lead oxide	0.269
For Sesqui-Silicates:		For Sesqui-Silicates:	
Lime	1.24	Lime	0.803
Baryta	3.40	Baryta	0.294
Magnesia	0.88	Magnesia	1.125
Alumina	0.76	Alumina	1.310
Ferrous oxide	1.60	Ferrous oxide	0.625
Manganous oxide	1.57	Manganous oxide	0.633
Lead oxide	4.95	Lead oxide	0.202

This table is to facilitate the study of slags that do not correspond to any of the typical slags but nevertheless give satisfactory results in the furnace.

Typical slags are definite combinations of silica, iron, lime, and sometimes alumina, the success of which has been proved and which therefore become representative. They fulfill the requirements of a good slag, which are, according to Eilers,† that it should not contain over 0.75% of lead or ½ oz. of silver to the ton, provided that the base bullion does not run higher than 300 oz., nor have a density over 3.6, nor permit accretions in the hearth, thus keeping the lead red-hot, nor any creeping up of over-fire.

* "Compendium der Metallurgischen Chemie," Bonn, 1882, p. 98.
† *Engineering and Mining Journal*, April 9, 1881.

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For instance, a typical slag discovered by Eilers is formulated thus:

$$6FeO,3SiO_2+2CaO.SiO_2.$$

In percentage it gives:

$$30.61SiO_2+55.10FeO+14.29CaO=100.$$

After deducting 10% for other ingredients of ore and fluxes, such as alumina, zinc oxide, etc., that cannot be brought under the head of ferrous oxide, like manganous oxide, or of lime, like baryta or magnesia, there remains:

$$27.55SiO_2+49.59FeO+12.86CaO=90,$$

which gives, in round figures,

$$28SiO_2+50FeO+12CaO+9Al_2O_3, \text{ etc.}=100.$$

Another slag, first brought into prominence by Eilers, has the formula,

$$6FeO.3SiO_2+4CaO.2SiO_2,$$

and therefore the following percentage of composition:

$$31.38SiO_2+45.18FeO+23.44CaO=100.$$

Deducting again 10%, there remains

$$28.24SiO_2+40.66FeO+21.10CaO=90,$$

which practical experience has altered to

$$30SiO_2+40FeO+20CaO+10Al_2O_3, \text{ etc.}=100.$$

The following slags have been thoroughly tested and are in successful use. The first seven (A to G) are those so designated by Iles in his paper already quoted.

Authority.	Type.	SiO_2.	Fe(Mn)O.	Ca(Ba,Mg)O.	Total.
Iles	A(a)	32	52	6	90
Iles	B(a)	35	45	10	90
Eilers	C	28	50	12	90
Iles	D(a)	34	34	16	84
Eilers	E	30	40	20	90
Schneider	F	33	33	24	90
Raht	G	35	27	28	90
Hahn	H	34	50	12	96
Page	I	33	36	16	85
Livingstone	J	30	36	20	86
Hahn	K	36	40	20	96
Iles	L	32	33	23	88
Murray	M	40	34	26	100

(a) No longer recommended by Iles.

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According to the ratio of FeO and CaO in the slag the expressions a quarter, a half, a three-quarter, a one-to-one slag, etc., are often used when

$$FeO : CaO = 4 : 1, = 2 : 1, = 4 : 3, = 1 : 1, \text{ etc.}$$

The slags in the table had been taken simply in an isolated way and not brought into a related order until Austin* tried to systematize their connection by plotting them. His

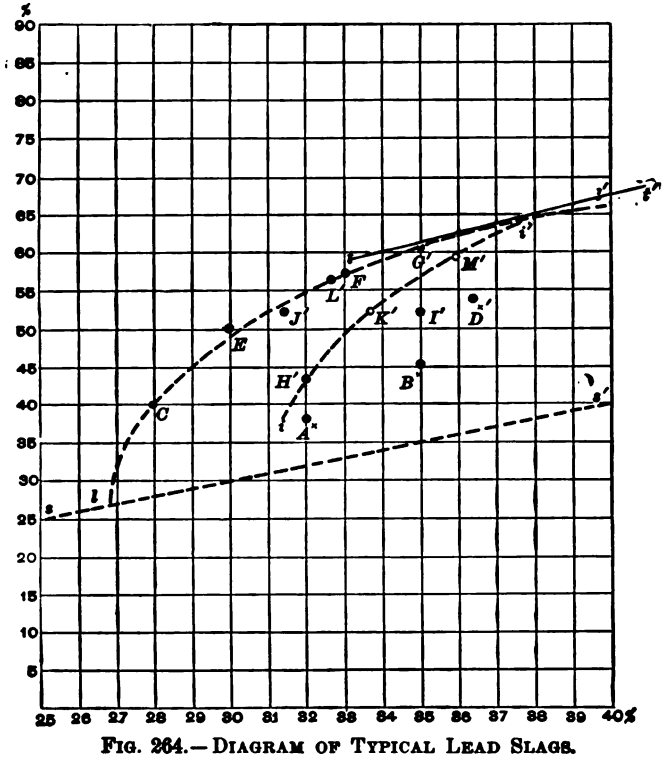


FIG. 264.—DIAGRAM OF TYPICAL LEAD SLAGS.

diagram is shown in Fig. 264. The straight line ss' represents the amounts of SiO_2, running from 25 to 40%, the upper curved line ll' the percentages of CaO and FeO, and the lower one ii' the same in a series running especially high in iron; tt' is a tangent parallel to line ss'. The capital letters with the (') have been multiplied with the proper coefficient to make their sums

* *Engineering and Mining Journal*, Jan. 27, 1894.

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add up to 90%, so as to permit their being plotted with the rest. The composition of slag *C*, *e.g.*, is read in the diagram as follows: Taking the ordinate running through the point C, the distance from the bottom to the intersection of line *ss'* represents the amount of SiO_2—28%—the distance from this to the point *C* the amount of CaO—12%—and the distance from *C* to the top the amount of FeO—50%. With exception of slags *A*, *B*, and *D*, which have fallen into disuse, and type *I*, which appears to belong to a separate class, all slags fit into two groups. Both curves show that with an increase of SiO_2 there is an increase of CaO, and that if from the point where CaO=FeO, *i.e.*, at SiO_2= 37%, a parallel *tt'* is drawn to the silica line *ss'*, the permissible variations in the relative proportions of FeO and CaO increase with the decrease of SiO_2, and lastly that with a high percentage of SiO_2 the totals are liable to add up to more than 90%, which means that highly silicious slags are better suited for clean, easy ores than those having many bases other than FeO and CaO. This is well exemplified in smelting Cripple Creek, Colo., gold ores rich in Al_2O_3, where the total of SiO_2, FeO and CaO does not often exceed 80%.

These typical slags are the general guides for the lead smelter. He chooses the one best suited to the character of his ore. He need not, however, adhere strictly to it. Slight variations are always liable to occur, and need give no uneasiness, if the furnace runs well and the slag analysis shows a low amount of lead and silver. At a high elevation, as at Leadville, Colo., 10,000 ft. above the level of the sea, a slag containing as much as 35% silica does not work well and one with a maximum of 32.5% must be chosen.

The old rule of making a slag with from 30 to 34% silica, an equal amount of metallic iron, and from 8 to 12% lime is practically obsolete.

In reviewing the table, it will be seen that the composition of the slags ranges as follows: SiO_2, between 28 and 36%; FeO, 24 and 52%; CaO, 6 and 30%.

Slags have been made that run as high as 40% silica, for instance, 40SiO_2, 40FeO, 10–12CaO, but they require much fuel, and are slightly viscid even then. The lowest practical limit in silica is probably reached with 28%. The percentages of ferrous oxide also represent extremes. As regards the lime, 30% is the

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highest Schneider* gives in his experiments, but 6% is too low. A modern American lead slag does not contain less than 10% of lime; European slags, as quoted by Kerl and Percy, run as low as 5%, although the tendency there is toward a higher percentage.

Physical Properties of Lead Slags.

The fusibility of a slag depends on the percentage of silica and the character of the bases; the effect of the latter will be discussed in the next paragraph. The more fusible a slag the larger will be the amount of charge smelted per unit of fuel. With the correspondingly low temperature in the furnace the reducing power will be diminished; therefore a very readily fusible slag is not always desirable.

The liquidity of a slag depends largely on its fusibility. A slag ought to be sufficiently liquid to allow a perfect separation of lead in the furnace and of matte in the slag-pot. A correctly composed slag, which would otherwise be liquid, becomes viscous if the weather suddenly becomes wet or cold; in such cases more fuel is required. It is difficult, even for the practiced eye, to say always whether the viscidity comes from lack of fusibility or from a slight insufficiency in fuel. As a rule, singulo-silicate slags containing earthy and metallic bases solidify quickly without first becoming pasty. When the tap-hole has been closed, the slag, if good, will drop slowly into the pot, drawing a small thread as it leaves the spout; a slag with an excess of base will fall off quickly in little round drops. Most slags have certain characteristics in their manner of running, which have to be studied by observation. In close connection with the running of the slag from the furnace is the manner of its rising in the slag-pot and the appearance of the surface when it has just solidified and is still red. Thus many slags show very characteristic surfaces.

Well-composed slags have a decided tendency to crystallize. The center of a cone of slag is generally more crystalline than any other part, because the cooling is slower. Slags that crystallize usually become amorphous if chilled suddenly, and cryptocrystalline if not given sufficient time to develop crystals. A small percentage of zinc oxide in the slag interferes greatly with

* "Transactions of American Institute of Mining Engineers," xi., p. 57.

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the crystallization. Iles thought at one time that the form in which a slag crystallizes stands in some relation to the percentage of lime it contains. He said (see Fig. 265):

"Slags with from 3 to 5% of lime crystallize like *a*; 8 to 12%, *a'*; 15 to 18%, *b*; 19 to 22%, *c*; 23 to 25%, *d*; 25 to 27%, *e*; 30 to 35%, *g*."

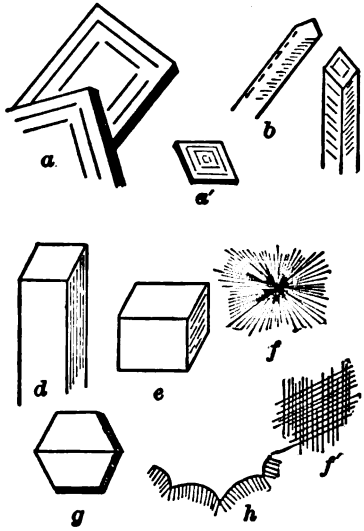


FIG. 265.—CRYSTAL-FORMS OF LEAD SLAGS.

The forms *f f' h* represent crystals formed in attempting to slag very silicious ores with lime alone. But the percentage of lime cannot be the only cause for the different forms of crystallization. Raht* obtained from a single pot of slag three different forms of large, well-developed crystals of approximately same composition.

	Globular Radiated Form.	Apparently Cubical Form.	Prismatic Form.	Liquid Slag.
SiO_2	36.50	36.60	37.1	37.20
Fe	20.30	20.00	20.4	22.10
Mn	2.10	2.20	2.0	1.80
Al_2O_3	7.80	7.40	7.2	n. d.
CaO	23.10	21.00	21.9	18.00
BaO	0.52	0.66	0.92	1.30
MgO	Trace.	Trace.	Trace.	n. d.
Pb	n. d.	n. d.	n. d.	1.50
S	0.40	0.36	0.55	n. d.

It is to be noted that the same slag will melt at a lower temperature when glassy than when crypto-crystalline.

* *Proceedings Colorado Scientific Society*, iv., p. 304, and private communication, April, 1897.

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There is some variety in the tenacity of slags. Silicious slags are generally tougher than those where the base prevails. The more crystalline a slag the more brittle. A pot of slag may show brittleness in one part and toughness in another. Slags are not always pure silicates as expressed in the chemical formula, they are often mixtures of different silicates or even chemical silicates holding indefinite compounds of silica and base in solution. If in these, crystals form, they represent the true silicate, being surrounded by a mass of an entirely different composition and different physical properties. This is important to remember in taking a sample for slag analysis.

The specific gravity of a slag is an important factor in its separation from the matte. Iles gives, as extreme figures, 3.3 and 4.16; as a common range for good slags, 3.4 and 3.6. The average specific gravity of 100 determinations of good slags gave him 3.691, and 3.6 is accepted as the highest specific gravity a good slag ought to have.

Slags do not possess to any extent the property of transmitting light. Single crystals are in exceptional cases transparent; sometimes they are translucent, but generally opaque.

Lead slags are usually black from their high percentage of iron. Thin pieces sometimes appear greenish-black; a large amount of iron will give a brownish hue. Lime produces a bluish or grayish tone. The luster of slags varies. It is occasionally metallic, but generally vitreous to resinous. Very often slags are dull.

Last may be mentioned the magnetic property of some lead slags, to which Iles first called attention. This is caused by the presence of magnetite* or magnetic sulphide of iron (Fe_3S_3). How the magnetic oxide gets into the slag is a matter for further investigation. Hahn suggests an incomplete reduction of ferric oxide; Guyard thinks it results from the oxidation† of metallic iron by lead oxide:

$$4PbO+Fe_3=Fe_3O_4+4Pb.$$

Neill‡ in examining a large number of lead slags running from 28 to 34% FeO for their magnetic properties found that they were not attracted by the magnet, while with high percentages, *e.g.*, Eilers' slag, SiO_2 28, FeO 52, CaO 12, this was the case; samples with 40% FeO were not attracted.

* Guyard, *Op. cit.*, p. 702.

† Berthier, "Traité des essais par la voie sèche," Liége, 1847, i., p. 354.

‡ "Transactions of American Institute of Mining Engineers," xx., p. 580.

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§ 69. Action of Fluxes and Influences of Foreign Matter.—The fluxes usually required are iron ore, manganese ore, limestone, dolomite, rarely fluorspar, and slag.

Iron.—An iron flux acts in three different ways. It gives a base for the silica in the ore,

$$FeO.SiO_2+FeO=2FeO.SiO_2,$$
$$4FeO.SiO_2+2FeO.2SiO_2=6FeO.3SiO_2.$$

Being reduced to metallic iron by means of carbon, it acts as a precipitating agent (§ 7),

$$4PbS+2Fe_2O_3+3C=4Pb+4FeS+3CO_2,$$
$$2PbS+4FeO.SiO_2+2C=2Pb+2FeS+2FeO.SiO_2+2CO.$$

It liberates lead oxide from its combination with silica, after which it is then reduced by means of carbon,

$$4FeO.SiO_2+2PbO.SiO_2+C=2Pb+4FeO.\ 2SiO_2+CO_2.$$

The larger the amount of iron, within reasonable limits, a slag contains the greater will be its fusibility and liquidity and the higher its specific gravity. Slags rich in iron are dangerous, as they are liable to cause the formation of crusts in the furnace. A slag high in iron is, however, a necessity if much zinc is contained in the ore, as iron favors the slagging and solution of zinc oxide. The purer an iron ore the greater will be its fluxing power, as only that part of the iron is available which is not required by its own silica, sulphur or arsenic. The silica not only limits the fluxing power, but also consumes limestone to form the slag. With the increase of slag in the blast-furnace the amount of lead ore of course diminishes, but the loss by scorification increases.

These remarks are equally applicable *mutatis mutandis* for the other fluxes.

ANALYSES OF IRON FLUXES. (*a*)

	Bruce Iron Ore, Colo.	Madonna Mine, Colo.		Bruce Iron Ore, Colo.	Madonna Mine, Colo.
Fe	66.44	42.05	SiO_2	2.39	5.10
Mn	0.01	3.12	CaO	0.12	.....
Zn	0.02	1.86	MgO	0.62	.....
Cu	0.02	.....	Al_2O_3	0.04	.....
Pb	.....	4.0–9.0	Ag, oz. p. t.	0.13	2.0–5.0
As	0.01	.....	Au, oz. p. t.	0.06	.....
S	.....	0.02			
Reference	(*b*)	(*c*)	Reference	(*b*)	(*c*)

(*a*) See also, "Leadville Carbonate Ores." (*b*) Guyard, *Op. cit.*, p. 647. (*c*) Dewey Bulletin No. 42, "United States National Museum," p. 47.

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*Manganese.**—The fluxing properties of manganese are similar to those of iron, and it may be substituted for the latter if desired. It makes a slag more fusible and liquid, so that with large amounts of manganese it is advisable to choose a slag that does not require much metallic base. The commonest manganese mineral is pyrolusite, MnO_2. Its oxidizing power has been said to be the cause that certain slags rich in manganese have so high a tenor in silver. This seems to be contradicted by the fact that slags rich in manganese have been made by Church† that ran low in silver and lead, the base bullion averaging 314 oz. to the ton.

SiO_2.	FeO.	MnO.	CaO.	MgO.	Al_2O_3.	Ag, oz.	Pb.
29.60	11.56	43.25	7.50	Trace.	6.34	Trace.	1.4
33.00	14.22	25.78	13.10	1.00	4.20	0.5	1.0

The oxidizing power of oxides of manganese on blende in the reverberatory furnace, when sulphide copper ores are smelted, is mentioned by Pearce,‡ who obtained a slag of SiO_2, 48%; MnO, 30%; and ZnO, 12.5%; some manganese (3%) also entering the matte (50 to 60% copper) as sulphide. Iles believes that in the blast furnace the amount of matte and speise formed diminishes with the increase of manganese in the charge.

Harbordt§ gives it as his experience that the percentage of matte formed is not affected unless the manganese is present in considerable quantities. Furman‖ say that with from 10 to 12% manganese dioxide in the charge the oxidizing power is noticeable. Another peculiarity of manganese is that it reduces the dissolving power of a slag for zinc oxide, magnesia, and barium sulphide.

According to Chisolm¶ 60,000 tons of manganiferous iron ores were smelted in Colorado in 1887, while in 1895** only 7,968 gross tons were produced.

* Iles, *School of Mines Quarterly*, v., p. 217, and *Engineering and Mining Journal*, March 12, 1881; Eilers, *Engineering and Mining Journal*, April 9, 1881; Penrose, "Manganese, Its Uses, Ores and Deposits," vol. i., Geological Survey of Arkansas, 1891; Church, "Transactions of American Institute of Mining Engineers," xxiv., p. 559.

† "Transactions of American Institute of Mining Engineers," xv., p. 612; *School of Mines Quarterly*, v., p. 322.

‡ "Transactions of American Institute of Mining Engineers," xi., p. 59.

§ Private communication, July, 1891.

‖ *School of Mines Quarterly*, xiv., p. 315.

** "The Mineral Industry," iv., p. 487.

¶ "Mineral Resources of the United States," 1887, p. 153.

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ANALYSES OF MANGANESE ORES AND MANGANIFEROUS IRON ORES.

	Montana. (a)	Arizona. (b)	Colorado. (c)
	Per Cent.	Per Cent.	Per Cent.
SiO_2	6.60	11 to 14	8 to 15
Fe_2O_3	3.20	4 to 9	35 to 50
Mn_2O_3	88.40	60	8 to 15
Ag, oz	9.00	.....	7 to 12

(a) Peters, "Mineral Resources of the United States," 1883-84, p 380. (b) "Transactions of American Institute of Mining Engineers," xxiv., p. 561. (c) Chisholm, "Mineral Resources of the United States," 1885, p. 348.

Lime and Magnesia.—The manner in which lime replaces iron in a slag can be expressed by changing the formula given above to

$$4FeO.2SiO_2+2PbS+2CaO+2C=2Pb+2FeS+(2CaO.SiO_2 +2FeO.SiO_2)+2CO.$$

The effect of lime is to decrease the fusibility and the specific gravity of slags. Slags rich in lime require more heat, *i.e.*, more fuel, a stronger blast, and consequently a higher furnace than if iron predominates. They generally give a good separation of slag and matte, and limestone being in nearly all cases cheaper than iron ore, the tendency in most large smelting works is to make slags rich in lime.

Schneider* found that with slags containing much lime less matte is formed than when they are rich in iron, and also that the matte is lower in lead and higher in silver. He explains it by saying that calcium sulphide is formed and then dissolved by the slag. Its presence in Leadville slags has been proved by Guyard.† The use of lime in slags is limited by foreign matter in the ore, and especially by the presence of zinc. In a general way it may be said that with zincky ores the less lime and the more iron a slag contains the better will the furnace work. It does not seem advisable to go beyond 16% of lime in a slag if from 9 to 12% of zinc is present in the charge. With 28% of lime the zinc simply refuses to enter the slag, but is volatilized and incrusts the furnace.

Burnt lime is rarely, if ever, used in the blast furnace. Bretherton‡ states that at the American Smelter, Leadville, Colo., he used for eight years burnt lime instead of limestone, with the

* "Transactions of American Institute of Mining Engineers," xi., p. 58.

† *Op. cit.*, p. 73.

‡ *Engineering and Mining Journal*, March 4, 1893.

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result that the furnaces not only put through more ore with the same labor and fuel than with limestone, but that cleaner slags were produced. Some of the leading Colorado lead-smelters experimented systematically for months with burnt lime and, finding no metallurgical or economic benefit, returned to the use of limestone.

Burnt lime was used many years ago in the iron blast furnace. The supposed advantages were that more ore could be put through per unit of fuel in a given time (as 56 CaO charged are equivalent to 100 $CaCO_3$); that the heat consumed by driving off the 44 CO_2 in the blast furnace was saved; and that the partial reduction of the 44 CO_2 by means of carbon did not occur. Grüner* calculated a saving of 10% of fuel by the use of burnt lime, while Bell† made it only 3.4%. The reason why burnt lime in the iron blast furnace has been given up is that it absorbs carbon dioxide and water vapor from the furnace gases, a heat up to redness‡ assisting the absorption of carbon dioxide, and lime parting even less readily with the combined water than with carbon dioxide. While the absorption generates the same amount of heat that the expulsion consumes, it occurs in the upper parts of the furnace, warming the gases that are passing out; meanwhile the lower part is being cooled by the expulsion, and this has to be made up by an extra amount of fuel. If lime is to be used, the lime must be burned in large pieces and the burning must be very complete and even carried to sintering, if possible (*i.e.*, with impure limestone), so as to diminish the absorption of carbon dioxide and water. The apparatus for burning lime must be easy to regulate and manage, so as to supply just the amount of lime required by the blast furnace, as storage and transportation for any distance are out of question. The fuel charged into the blast furnace must be dry, and the ore and iron flux not only dry, but free from carbon dioxide. Finally, the height of the furnace may have to be increased to prevent the furnace gases from passing off at too high a temperature. The many difficulties in the way of using burnt lime have deterred the iron smelter from it, and special economic conditions must prevail to make it profitable to the lead smelter, as for example, when on account of

* *Annales des Mines*, 1871, xx., p. 325; and *Dingler's Polytechnisches Journal*, 1872, cciv., p. 309.

† "Chemical Phenomena of Iron Smelting," London, 1872, p. 138.

‡ Bell, "Manufacture of Iron and Steel," London, 1884, p. 60.

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high freight charges burnt lime can be delivered at the smelter at the same cost as limestone. The use of burnt lime in the iron blast furnace has lately been again discussed by iron metallurgists.*

Magnesia is undesirable in a lead furnace, as it makes a slag pasty and streaky, but in many cases the only available limestone is dolomitic, and it must be made the best of. This undesirable property of magnesia is especially observable if the slag contains zinc. Magnesia and zinc oxide appear to intensify each other's property of being difficult to slag. In a slag containing 8% zinc and from 2 to 3% baryta, very common just now in Colorado, from 2 to 3% magnesia shows a decidedly bad effect, and 5% causes a great deal of trouble and is liable to chill. This is perhaps to be explained by the fact that the silicate of magnesia has a higher specific heat than the silicate of lime and ought to be corrected by more fuel.

Magnesia is generally figured in a slag as replacing lime; it may be doubted whether this is justifiable (Harbordt).†

ANALYSES OF LIMESTONE.

	Cañon City, Colo.	Glass Mine, Colo.	Carbonate Mine, Colo.	Iron Co., Mo.	St. Joseph, Mo.	Mine La Motte, Mo.
$CaCO_3$	88.90	49.57	57.95	47.50	60.34	55.2
$MgCO_3$	6.30	37.08	39.65	42.19	32.17	40.9
SiO_2	3.10	4.23	0.76	5.11	3.77	1.27
$(FeAl)_2O_3$	1.50	.....	.....	Fe_2O_3 4.67	6.88	.....
$FeCO_3$	.....	6.23	.....	.....	.....	.....
Reference	(a)	(a)	(a)	(b)	(c)	(d)

(a) Guyard: Emmons, *Op. cit.*, p. 646. (b) Gage, "Transactions of American Institute of Mining Engineers," iii., p. 117. (c) Monell, *School of Mines Quarterly*, ix., p. 214. (d) Neill, *Ibid.*

Fluorspar is of slight importance in smelting lead-silver ores in the blast furnace. It forms with barium and calcium sulphate readily fusible compounds, and assists in fluxing zinc either as sulphide or oxide. This is mainly due to the fact that fluorspar, when once melted, is very liquid, and thus assists other less fusible compounds to enter the slag. Its chemical action in volatilizing silicon as silicon fluoride need not be taken into consideration. Foehr‡ makes a great many claims for fluorspar; that the use of from 1 to 5% in roasting ores in a reverberatory furnace saves fuel, that adding it to the charge in refining lead prevents shots of lead from being retained by the litharge, etc.

* *Journal Iron and Steel Institute*, Bell, 1898, ii., p. 274; 1894, ii., p. 38; Howden, 1894, i., p. 85; Cochrane, 1894, ii., p. 62; Howe, 1894, ii., p. 87; *Stahl und Eisen*, Kosmann, 1891, p. 311.

† Private communication, July, 1891.

‡ *Engineering and Mining Journal*, June 21, 28, 1890.

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Slag.—There are four reasons for the use of slag in blast furnace charges: (1) It may contain too much lead or silver to be thrown away. (2) It makes the charge less dense. (3) It helps the actual smelting process because the slag having been already melted, it will re-melt easily and promote the smelting of the ore itself; and (4), if it be more acid or basic than the slag that is being formed by the smelting mixture, it will act as an acid or basic flux.

With a furnace running in a normal way, some rich slag is always produced. This is especially the case when the last slag in the furnace is being tapped and the blast passing through the tap-hole blows out valuable parts, which enrich the slag in the pot (blow-pot). Then again, when much matte comes out with the slag, it is liable not to settle out perfectly.

With a coarse, easy charge the addition of slag is not absolutely necessary, although some is always given—say from 150 to 200 lb. for 1,000 lb. of ore. With fine ore the addition of slag is necessary, as otherwise the blast would not penetrate evenly through the smelting mixture, but form blow-holes. As much as 25% of the charge may have to consist of slag. With ores rich in zinc, slag is added to increase the fusibility to the extent of 20%.

Slag more basic than the normal slag comes into play where, for example, matte is being concentrated in a reverberatory furnace, and the resulting slag contains much iron that is available for the blast furnace.

In smelting the by-products of desilverizing works in the blast furnace without ore, the amount of slag added goes up as high as 50%.

Alumina.—From the composition of the typical lead slags it will be seen that the place alumina occupies in lead smelting is generally a subordinate ore. When it is present in large quantities it becomes a question whether it acts as an acid or a base. It is known in a general way that with a high percentage of silica, alumina acts as a base; with a low percentage it acts as an acid. Iles* gives this as his experience in lead smelting. Hahn,† however, thinks that alumina always acts as a base, and says that an

* "Mineral Resources of the United States," 1883-84, p. 433.
† *Ibid.*, 1882, p. 338.

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increase of alumina requires also an increase of silica, or, what would be the same, a decrease in the bases.

Schneider* found that as a general rule an increase of alumina called for an increase in the proportion of lime. This may mean a decrease in silica, the alumina acting as an acid.

Another interpretation is that the presence of alumina calls for a diminution of iron, the percentage both of silica and lime remaining high. This agrees with the inference drawn from Austin's diagram (Fig. 264) that an increase of lime calls for an increase of silica, and many metallurgists work out from this basis. Others, however, leave the iron intact and cut down both silica and lime. If with a slag of the composition: SiO_2 36, FeO 33, CaO 15, an ore running high in alumina is to be smelted, the slag would be changed to

$$SiO_2\ 30,\ FeO\ 33,\ CaO\ 12 + 1.6\ (\text{equivalent to } 3\ Al_2O_3) + Al_2O_6\ 6\ (\text{replacing } SiO_2\ 6),$$

and do satisfactory work.

Howe,† summing up the statements of Hahn and Schneider, suggests that the part played by alumina may depend upon the proportion of the other two fluxes, lime and iron, and that in calcareous singulo-silicates, low in iron, alumina may act as an acid and in ferruginous slags, low in lime, as a base. The idea seems to be confirmed by the experience of Peters‡ in smelting Mount Lincoln§ ores in Colorado. The writer's own impression is that alumina acts as an acid in ferruginous slags, as in smelting an ore resembling an argentiferous clay he could not get his furnace to work with a slag that ran higher than 28% silica.

Henrich,‖ in a very interesting paper on smelting sulphide copper ores rich in silica, alumina, magnesia, and low in iron, came to the conclusion that alumina always acts as an acid, and recommends two general types of silicate-aluminates for ores rich in alumina, on the hypothesis that $2Al_2O_3$ are equivalent to $3SiO_2$.

Type I. { 8 metallic oxide / 4 earthy oxide } : { $2Al_2O_3$ / $3SiO_2$ } Mono-Silicate-Aluminate.
12 oxygen of base : 12 oxygen of acid = 1 : 1.

Type II. { 8 metallic oxide / 4 earthy oxide } : { $2Al_2O_3$ / $9SiO_2$ } Bi-Silicate-Aluminate.
12 oxygen of base : 24 oxygen of acid = 1 : 2.

* "Transactions of American Institute of Mining Engineers," xi., p. 57.
† *Engineering and Mining Journal*, Nov. 17, 1883.
‡ *Ibid.*, Nov. 24, 1883.
§ "Transactions of American Institute of Mining Engineers," ii., p. 310.
‖ *Engineering and Mining Journal*, July 17, Aug. 28, Oct. 2, 1886; Dec. 27, 1890.

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The theoretical composition of these two types is:

	I.	II.
SiO_2	16.1	35.0
Al_2O_3	18.2	13.2
FeO	25.7	37.3
CaO	40.0	14.5

By a combination of them he thinks it possible to meet all cases where the percentage of alumina is high. Stone,* Elbers,† and Kosmann‡ have discussed the matter from the iron-metallurgist's point of view.

To sum up: the question, under what conditions alumina ceases to be a base and becomes an acid, is not settled. It has to be determined for each slag by synthetical experiments in the laboratory.

Barite.—Barium sulphate in the blast furnace is partly converted into barium sulphide and partly into barium silicate. In both forms it causes an imperfect separation of matte and slag. The reaction is usually expressed by the following formula:§

$$2BaSO_4+8Fe+4SiO_2=(BaS.FeS)+7FeO.BaO.4SiO_2.$$

This would account for the formation of barium sulphide, even if no barium sulphate were reduced by means of carbon. It would also lead one to suppose that all barium sulphide entered the matte, but the fact is that, while a very little can be discovered in the matte, most of it is dissolved by the slag. Further, the proportions given do not correspond to the results obtained in a lead furnace. That barium sulphide can enter the matte under suitable conditions is proved by the mattes made in the Altai Mountains, which contain, according to Jossa and Kurnakoff,‖ from 4 to 22% barium.

According to Schweder¶ there is another and more important reaction:

$$3BaSO_4+4SiO_2+FeS=3BaSiO_3+FeSiO_3+4SO_2$$
$$BaSO_4+2SiO_2+Fe=BaSiO_3+FeSiO_3+SO_2.$$

* *Engineering and Mining Journal*, Nov. 24, 1883.
† *Ibid.*, March 27, July 31, 1886.
‡ *Stahl und Eisen*, 1892, p. 270.
§ Balling, "Metallurgische Chemie," p. 89.
‖ *Berg- und Hüttenmännische Zeitung*, 1886, p. 547.
¶ *Ibid.*, 1879, p. 38; *Iron*, 1879, xiii., p. 387.

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This means that in the presence of a metallic sulphide or a metal, barium sulphate is readily decomposed by silica; the liberated sulphur trioxide is decomposed at the elevated temperature into sulphur dioxide and oxygen, and this has an oxidizing effect on the iron sulphide or the metallic iron.

Schweder found in his experiments that calcium sulphate and sodium sulphate act in the same way.

Silicates of iron and baryta form very liquid slags.

In computing a charge the baryta is figured as replacing part of the lime.

From what has been said, it will be clear that all the sulphur contained in the barite cannot be figured into the charge. In fact, if 10% of the sulphur is figured as combining with iron to form iron sulphide, the amount will be covered.

Pyrite.—If an ore containing pyrite be smelted directly in the blast furnace, it will consume iron as shown by the formula:

$$FeS_2+Fe=2FeS.$$

In calculating a charge this iron has to be added. There are, however, cases in smelting oxidized ores when part of the sulphur passes off as sulphur dioxide and comparatively little matte is made. Henrich* gives his successful experience in smelting carbonate ores at Benson, Ariz., consisting of galena (15%) and anglesite (75%), with silver-bearing pyrite. He obtained very little matte (20 lb. from 13 or 14 tons of ore), but considerable sulphur dioxide. This he explains as having been caused by the following reactions:

$$2FeS_2+5PbSO_4+SiO_2=5Pb+Fe_2SiO_4+9SO_2.$$
$$PbSO_4+PbS=2Pb+2SO_2.$$

He says that the furnace ran rapidly and became very hot, so that the fuel (coke) had to be cut down from 12.5 to 11%, the pressure of the blast being $1\frac{3}{8}$ in. mercury.

Chalcopyrite and Copper Ores.—In the blast furnace the aim is always to carry the copper into the matte, which it enters as cuprous sulphide. Copper, having a greater affinity than any other metal for sulphur, will generally take up all the sulphur in the charge to form Cu_2S, and what is left is then available for iron, lead, etc. If a charge does contain copper and not enough sulphur to form cuprous sulphide, some copper will be reduced

* *Engineering and Mining Journal*, Sept. 22, 1883.

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to metal and be alloyed with the lead. The alloy hardens, sinks to the bottom, and closes up the passage of the lead-well. Such an alloy contained Cu 35.1% and Pb 47% and Ag 170 oz. per ton (Iles). There is one case where even with sufficient sulphur to form cuprous sulphide the copper combines with the lead. This is when matte is concentrated in the blast furnace with a highly ferruginous slag. The affinities of copper and iron for sulphur and silica seem to become disturbed.

If the slag be too basic the iron takes up some sulphur and goes into the matte instead of separating out and forming a crust; thus some sulphur belonging to the copper may be taken away, and this alloys with lead. Another way of explaining the fact would be that reactions between sulphides and oxides of copper take place similar to those of the reverberatory furnace, and the resulting metallic copper becomes alloyed with reduced lead. Whatever may be the theory, the fact remains that any excess of iron has to be avoided in the slag if the copper is to be concentrated in the matte and not partly driven into the lead. This is liable to occur when the matte contains about 12% of copper, and increasingly so with the increase of copper.

Blende and Zinc Oxide.—Zinc is the metal that causes the greatest difficulty in the lead blast furnace in whatever form it may occur. Blende* is decomposed by iron oxides and silicates, the resulting zinc oxide entering the slag; metallic iron liberates metallic zinc, but most of the zinc sulphide entering the blast furnace remains undecomposed and enters the matte as well as the slag. Iles† avoids using scrap iron if he has blende in the charge. Generally the percentage of zinc found by analysis in matte and slag will be about equal. Blende makes matte and slag less fusible, obstructs the separation, and carries other metallic sulphides into the slag. With ores containing little zinc this imperfect separation can be remedied by an addition of chalcopyrite‡ to the charge. If blende is present to any considerable extent, the ore must be roasted before smelting, to form zinc oxide, although the operation is connected with a loss in silver, from 1 to 2 oz. per ton, according to the percentage of zinc. In order to carry the oxide off, the slag has to be very

* Plattner, *Berg- und Hüttenmännische Zeitung*, 1854, p. 81.
† *School of Mines Quarterly*, xviii., p. 18.
‡ Hahn, "Mineral Resources of the United States," 1882, p. 343.

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fusible, as zinc silicate proper is infusible, and thus lowers the fusibility of any other silicate, and retards the smelting. It is questioned by some metallurgists whether the generally accepted statement that zinc is present in the slag as a silicate is a correct one. They maintain that most of it is held in igneous solution as oxide, some as sulphide, and perhaps a very small amount as silicate. A partial evidence of the first belief is that if a piece of charcoal be plunged into a pot filled with slag, after the matte has been allowed to settle out, white fumes of burning zinc will pass off from the surface. While the zinc from zinc silicate is completely reduced by carbon at a white heat, it is not known at what temperature the reduction begins. That zinc is present in the slag as sulphide is readily seen when zinc-bearing slag is tapped or poured. Evidence for the presence of zinc silicate is lacking, as far as the writer is aware. If zinc oxide is to be slagged or dissolved care must be taken that it is not reduced to metal, hence the smelting has to be done quickly and at a low temperature. This requires a slag not high in silica and with a preponderance of iron. If, however, a slag high in lime has to be used, it has become the practice to figure one-half of the zinc (ZnO) as replacing an equivalent amount of lime (CaO) in the slag. If this is done, it will be found that the total of SiO_2, FeO and CaO will have to be made larger than the usual figure of 90. The amount of lime that the slag may retain to work well in the furnace then depends mainly on the percentage of matte that is formed; the higher this is the lower must be the lime. In any case, in ore smelting, a slag ought not to contain more than from 6 to 8% of zinc, as otherwise there is too great a loss in metal. Slags are made that contain more than 8% zinc, but that this is commercially good practice few will maintain. The subjoined slags* are said to have been made with satisfactory results:

SiO_2.	FeO.	CaO.	ZnO.
34	34	17	9
35	38	13	12

Canby† and Weinberg‡ publish a number of zinc-bearing slags high in lime made successfully by them. The first table is by Canby, the second by Weinberg:

* *Engineering and Mining Journal*, Aug. 29, 1896.
† *Ibid.*, Sept. 26, 1896. ‡ *Ibid.*, Dec. 19, 1896.

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SiO_2.....	27.9	30.0	26.0	27.0	24.5	26.0	27.0	26.4	30.0
FeO.....	33.9	29.0	33.4	31.5	29.4	32.1	26.5	22.7	24.5
CaO.....	14.8	14.0	14.4	19.0	24.5	19.0	24.8	24.8	22.8
ZnO.....	16.6	15.5	19.8	15.0	14.5	15.0	14.1	21.0	17.7

SiO_2.	FeO.	Al_2O_3.	CaO.	PbO.	Cu.	Ag.	ZnO. (a)	ZnS. (a)
31.96	31.82	8.77	6.80	0.24	Trace.	Trace.	13.84	6.71

(a) Calculated.

The slag produced in smelting zinc-bearing ores at the "Herzog Julius and Frau Sophieen" Works in the Harz Mountains* has the following composition: SiO_2 16.90, FeO 35.05, ZnO 19.64, $BaSO_4$ 10.24 (=BaS 8.13), Al_2O_3 6.31, CaO 6.95.

It is to be noted that all these slags were made in blast furnaces in which lead and matte were tapped from the bottom of the crucible. It would hardly be possible to make them in a furnace having an Arents' siphon tap, as the crucible would be quickly crusted over. If zinc oxide be reduced to metal in the lower parts of the furnace by carbon, or to a slight extent by metallic iron, it becomes volatilized. The vapor ascending carries with it lead and silver, and, being oxidized higher up by carbon dioxide or water-vapor or by the oxygen taken from any readily reducible oxide, carries off metal as fluedust, and forms accretions on the sides of the furnace, beginning about 7 ft. above the tuyeres and growing in thickness up to the charging-door. With the use of a blast of very high pressure† this evil has been diminished, at times entirely avoided. Nevertheless the loss in lead and silver must remain the same.

If zinc oxide, not reduced to metal on its downward course in the charge, comes, with the presence of carbon, in contact with lead sulphide or sulphate, it is converted into sulphide; if sufficient iron is present it will decompose the lead sulphide and the zinc oxide will remain unchanged.

The aim must therefore be to remove as much zinc as possible from the ore before it is smelted. The various attempts to do this have not been very successful, with the exception of mechanical concentration, and this with very fine-grained ores is out of place on account of the intimate contact of blende with galena; with oxidized ores the losses in precious metal would be too great.

* *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Pruessen*, 1877, xxv., p. 148.

† Kloz, *Engineering and Mining Journal*, April 10, 1897.

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A number of plans for removing zinc from ores not suited for concentration have been tried or suggested. Thum* gives a way of distilling the roasted ore in inclined cylindrical retorts, the zinc vapors being carried off at the upper end and the lead compounds and residue removed at the lower. Lumaghi† proposes to work on a similar plan. Binon and Grandfils‡ describe a furnace with vertical retorts, which are charged from the top and discharged at the bottom, while the zinc vapors are drawn off by a horizontal condenser near the charging opening. Chenhall§ suggests a similar method. Another dry method is that of Simmonet,‖ who roasts finely pulverized lead and silver-bearing blende with coarse-crushed limestone. Most of the lead and silver pass into the limestone, which has been in part changed to caustic lime, calcium sulphate, and sulphide, and can be separated from the fine zinc ore by screening. The writer once experimented with this method on several kinds of ore, but too little silver passed into the limestone to make it a success. The method is, however, worth trying, as it succeeded with Simmonet, and may again with suitable ore. The F. L. Bartlett process for treating complex zinc-lead ores has been fully described in § 53.

One class of wet methods aims to convert the zinc sulphide into zinc sulphite and sulphate, and to leach these with or without the addition of sulphuric acid. This is represented by Parnell,¶ Chenhall,** Siemens and Halske,†† Croselmire,‡‡ Lowe,§§ West,‖‖ and Létrange.¶¶ In the Lower Harz*** (Prussia) the old process of heap-roasting the mixed sulphide ores and leaching out the zinc sulphate has received a fresh impetus from the manfacture of a white paint, called "Lithopon," which is a mix-

* *Berg- und Hüttenmännische Zeitung*, 1875, p. 1.

† *Engineering and Mining Journal*, Aug. 23, 1890.

‡ *Revue Universelle des Mines*, 1879, vol. v., p. 229.

§ *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, 1880, p. 462.

‖ *Annales des Mines*, 1870, vol. xvii., p. 27.

¶ *Berg- und Hüttenmännische Zeitung*, 1881, p 254; 1883, p. 242.

** *Proceedings of the Institute of Civil Engineers* (England), 1883, lxxiv., p. 229; *Berg und Hüttenmännische Zeitung*, 1884, p. 465; *Iron*, xxii., p. 465; *Chemical News*, xlii., p. 201.

†† *Wagner's Jahresberichte*, 1888, p. 368.

‡‡ *Engineering and Mining Journal*, Sept. 29, 1888; Feb. 9, 1889.

§§ *Ibid.*, Jan. 5, 1889, p. 17.

‖‖ *Ibid.*, March 14, 1891.

¶¶ *Berg- und Hüttenmännische Zeitung*, 1882, p. 489; 1883, p. 287.

*** *Zeitschrift für Berg- und Hütten- und Salinen Wesen in Preussen*, xxv., p. 144; *Berg- und Hüttenmännische Zeitung*, 1890, p. 131; 1891, p. 184; "The Mineral Industry," 1895, iv., p. 88.

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ture of barium sulphate and zinc sulphide resulting from the double decomposition of barium sulphide and zinc sulphate.

The other method of making zinc soluble is to convert it into chloride. Maxwell-Lyte* and Wilson† have based their processes on hydrochloric acid as a solvent. Electrochlorination has been tried by Slater.‡ Later suggestions for the treatment of zinc-bearing lead ores will be found in the annual reviews by the writer in "The Mineral Industry," II., 431; III., 431; IV., 472, V., 404, headed "Recent Improvements in Lead Metallurgy," and the reviews on the metallurgy of zinc.

In smelting zinc-bearing by-products from refining works in a blast furnace with Arents' siphon tap and with little or no ore the same slag is run over and over again until it becomes saturated with zinc. When a ferruginous slag contains 10% of zinc oxide it begins to be very mushy, and with 16% a gritty mass forms on top of the lead in the furnace, and the crucible cannot be kept open for any length of time. As far as the writer's experience goes, 16% is the maximum amount of zinc oxide that a singulo-silicate slag rich in iron can bear in a furnace having an Arents siphon tap.

In zinc smelting where the zinc is reduced and distilled in an atmosphere of carbon dioxide in small retorts, the conditions are different from those which prevail in lead smelting in a blast furnace. The chemical behavior in the retort of zinc oxide mixed with impurities ordinarily present in ores has been investigated by Prost.§ He finds that the reduction of the oxide is unaffected by presence of Fe_2O_3, Al_2O_3 and PbO, taking place at 1075° C. in case of pure ZnO, and at 1200° to 1250° when mixed with the other oxides in equal molecular proportions. Zinc sulphide is decomposed by iron, the action at any given temperature depending upon the proportion of iron, which should always be in excess of the amount theoretically requisite for combination with the sulphur. With a mixture of zinc sulphide and 190% excess of iron, the zinc was wholly volatilized at 1200° to 1250 C., while at 1075° considerable remained in

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* *Engineering and Mining Journal*, March 10, 1888.

† *Chemiker Zeitung*, 1884, p. 167.

‡ *Engineering and Mining Journal*, June 2, July 14, 1888.

§ *Bulletin de l'Association Belge des Chimiste*, x. 6, pp. 246-268; *Journal of the Society of Chemical Industry*, 1896, p. 813.

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the residue. Lime proved more efficient than iron, an excess of 75% over the theoretical quantity decomposing the zinc sulphide completely at 1250° C., while with sufficient lime the same result can be obtained at the melting point of gold (1200° C.). These experiments were made with a chemically pure, crystalline sphalerite.

At 1200° to 1250° calcium sulphate exerted no retarding influence on the reduction of zinc oxide, even in presence of lead and iron silicates, which might convert reduced calcium sulphide to silicate and facilitate the formation of zinc sulphide. Magnesium sulphate, which at a high temperature is decomposed by carbon into magnesia and sulphur dioxide, favors the production of zinc sulphide.

Experiments with a mixture of ZnO, PbO, and SiO_2 heated together at 1000° C. and reduced by carbon at 1200° to 1250° C., indicated the formation of a double silicate irreducible at that temperature, but completely reducible at 1450° to 1500.°

Antimony.—Antimony occurs either as a sulphide or an oxide. The antimony sulphide behaves on the whole like lead sulphide, but is much more volatile. If decomposed by metallic iron the resulting metal is more likely to combine with the lead than it is to form a speise with any excess of iron that may be present. It may also be volatilized. The oxide is generally present as an antimoniate of lead or iron, and this, being reduced to an antimonide, combines with the lead or the speise, if any is made, or with the matte. The two main injurious effects of antimony, therefore, are that it causes loss by volatilization and impairs the character of the lead. Antimonial speise is rare, and in making up an ore-charge no account need be taken of the small quantity of iron likely to be consumed by the antimony. It forms occasionally in smelting antimony skimmings (see § 128).

Two difficulties have to be contended with in treating the antimonial by-products of refining works in the blast-furnace. If the slag contains but little iron, thus requiring a high temperature, much antimony and lead are volatilized; if rich in iron, some speise is liable to form, which either separates out, causing the loss of the antimony, or becomes mixed with the slag, making it rich. A ferruginous slag is generally preferred as the lesser evil of the two.

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Arsenic.—Arsenic occurs very frequently in argentiferous lead ores and must not be neglected in computing a charge. It causes loss by volatilization and combines with the lead, but not to such an extent as antimony, as it has a great affinity for iron and is liable to form a speise. Arsenic combines with iron in various proportions, from Fe_2As on. Speise having a composition that lies between Fe_2As and Fe_4As is fine grained; it is not readily fusible, and not liquid when melted; it is likely to form hearth accretions ("speise sows") and to prevent a good separation of lead. Speise of the composition Fe_5As has large cleavage planes, similar to spiegeleisen; it is moderately fusible and pretty liquid when melted, does not readily form accretions, and retains only very few shots of lead. A speise of the composition Fe_6As and upward seems to be just as harmful as one that contains less iron than Fe_5As. In computing a charge some metallurgists add enough iron to form Fe_4As, keeping probably in mind that some arsenic is volatilized or combines with the lead; others assume Fe_5As. In making up a charge, if the ore does not contain too much arsenic, it is possible to avoid producing any speise at all by assisting part of the arsenic to pass off with the gases and the rest to combine with the lead and enter the matte. This can be done by making the charge open, cutting down the fuel and diminishing the reducing action, and by cutting down the lime. The formation of speise can be controlled by the lime in the charge. The probable explanation for this is that iron has a greater affinity for silica than for arsenic. If there is enough lime present to satisfy the requirements of silica for forming a fusible slag, some of the iron can combine with the arsenic and form a speise; if not, silica combines with all the iron and the arsenic is partly volatilized, the rest combining with the lead or entering the matte. If there is so much arsenic present that the formation of speise cannot well be avoided, a slag should be selected that requires a high formation temperature.

By taking a readily fusible slag the speise, floating on the lead beneath the matte and far removed from the hottest part of the furnace, is liable to chill. Thus the fusibility of the slag must stand in inverse ratio to the amount of speise formed. With large amounts of spiese even a sesquisilicate slag may be necessary.

Sometimes old iron castings are added to loosen up any speise sows that have been formed in the hearth. This is useful if the

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speise does not contain enough iron, as it supplies a deficiency, but does no good if the accretion is caused by too low a temperature.

§ 70. Fuels Used in the Blast Furnace.—The fuels commonly used in lead smelting are coke, charcoal, and a mixture of the two. Occasionally part of the coke is replaced by bituminous coal, and experiments have been made to substitute anthracite for part of the coke; gaseous fuel has also been tried.

Coke.—The coke to be suited for smelting purposes ought to be hard enough to bear the burden, slightly porous and low in ash. As regards the strength of the cell walls, Connellsville coke is the best, gas coke the worst. But Connellsville coke is very dense, and consequently does not oxidize readily. In order to overcome this, a high pressure of the blast is needed, which is on the whole not desirable. With the higher pressure in common use now, the coke may be more dense than was permissible a few years ago. Gas coke might form a suitable fuel as regards its porosity, if the pores were not too large; they become partially clogged and the lumps of coke become glazed so as to be not only useless but harmful. A coke, therefore, that combines strength with a moderate degree of porosity is what is wanted. The disadvantages of Connellsville coke are, however, more than made up for by the fact that it is not much broken up in transportation and by its low percentage in ash and its uniform quality. Thus, if coke has to be brought from a distance, Connellsville coke is the best.

The amount of ash in coke varies from 10 to 20%; 1 cu. ft. of dry coke weighs about 50 lb.; it has about 50% of cell-space, and can bear about a 90-ft. charge without crushing.

In computing a charge the ash of the coke has to be taken into account.

Coke, used alone as fuel, gives clean slags and little fluedust.

COMPOSITION OF SOME COKES USED IN LEAD DISTRICTS.

	Connellsville, Pa.	Cardiff, Wales.	Grand River, Colo.	El Moro, Colo.	Crested Butte, Colo.	Pocahontas, Va.
Fixed carbon	87.46	95.00	93.75	87.47	92.03	93.550
Moisture	0.49	0.01	.....	.....	.....	0.347
Ash	11.32	4.26	5.49	10.68*a*	6.62*a*	5.749
Sulphur	0.69	0.68	0.76	0.85	.....	0.597
Phosphorus	0.029	0.02	0.10	.....	.....	.....
Volatile matter	0.011	.....	.....	1.85*b*	1.35*b*	0.757
Reference	(*c*)	(*c*)	(*c*)	(*d*)	(*d*)	(*f*)

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ANALYSES OF COKE ASH.

	Connellsville, Pa.	Starkville Colo.	Grand River, Colo.	El Moro, Colo.	Crested Butte, Colo.	South Park, Colo.
SiO_2	44.64	65.45	50.1	84.50	42.25	29.10
Al_2O_3	25.12	25.60	.....	8.40	23.07	23.10
Fe_2O_3	23.73	8.00	14.3	7.10	26.42	47.80
CaO	6.95	.....	.....	.....	3.50	.....
MgO	1.91	.....	.....	.....	1.40	.....
Reference	(*d*)	(*h*)	(*h*)	(*e*)	(*g*)	(*e*)

(*a*) Generally higher. (*b*) Including moisture. (*c*) "Mineral Resources of the United States," 1887, p. 396. (*d*) "Tenth Census of the United States." 1880, xv., p. 72. (*e*) Emmons, "Geology and Mining Industry of Leadville, Colo.," p. 642. (*f*) "Transactions of American Institute of Mining Engineers," xix., p. 1084. (*g*) Private communication by C. H. Livingstone. (*h*) Private communication by M. W. Iles.

Before the coke is fed into the furnace all the fine parts have to be removed. This is done with a coke-fork having prongs ½ in. apart, while removing the coke from the sheds. Sometimes a scoop is used and the fines screened out by dumping the coke over a grizzly, which discharges the coarse coke on the feed-floor and the fines into a bin, whence they are removed, usually to be burned under the boiler, occasionally in the gas producer of the roasting furnaces.

Charcoal.—As regards porosity charcoal is the best fuel for a lead furnace, as it consists* of a large number of small cells joined to each other by porous walls. Hence, being readily oxidized, it is a good reducing agent for oxidized ores, and does not require a high blast, which is an advantage. Its greater porosity causes greater bulk (3 : 1), thus making the charge looser, which is favorable for quick smelting; further, its percentage of ash is low. The great disadvantage of charcoal is that very few kinds can bear any heavy burden; it breaks up and crumbles. Fine charcoal is not only worthless as a fuel, but it is a bad conductor of heat. It makes unclean slags and also causes loss in metal by increasing the amount of fluedust, hence nobody uses charcoal alone unless absolutely forced to do so.

Nut-pine (piñon) charcoal is the best, but it has to be well burned. Charcoal from lighter woods, such as yellow and white pine, quaking aspen, and cottonwood cannot be used alone in the furnace, and even with coke only a small percentage is allowable; some metallurgists condemn it entirely. Mesquite makes a good charcoal, but it is obtained with difficulty in large pieces. Charcoal from hard woods, such as mahogany, cedar, and oak, de-

* Thörner, *Stahl und Eisen*, vi., p. 71.

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crepitates in the furnace. When charcoal is exposed for any length of time to the open air it breaks up and the amount of fines becomes large. On the other hand, its quality is said to be improved by storing, through the oxygen that it absorbs. Lead smelters do not like to have large amounts of charcoal on hand. It should be stored where it is not exposed to the sun. Charcoal has in a few instances been replaced by an equivalent in reducing power of split wood cut up into 12-in. lengths. The wood keeps the charge open and the charcoal formed arrives less crushed at the tuyere-region than if it had been charged in that form. Lang* advocates its use in matting in the blast furnace, having replaced with it one-half of the coke.

Lightwood charcoal contains about 2% of ash, consisting principally of alkali and alkali-earth carbonates, and does not affect a charge to any appreciable extent. One bushel weighs about 14 lb. The height of charge it can bear varies too much with the different kinds of charcoal to give a general figure; it is less than with coke.

Coke and Charcoal.—From what has been said it is obvious that the ideal fuel for the lead smelter must combine the strength of coke and the porosity of charcoal; thus, at least with a low-pressure blast, a mixture of coke and charcoal will put through more charges in a given time than either alone. The coke bears up the charge and prevents the charcoal from being crushed. This burns quickly, helps to ignite the coke, and, having hardly any ash, leaves hollow spaces for the blast to penetrate. At present when furnaces are run with a pressure of from 2 to 2½ lb. to the square inch and oxide ores form a small percentage of the charge, charcoal is used very rarely.

Coke and Bituminous Coal.—Neill† succeeded in 1891 in replacing part of the coke by a non-caking or only slightly-caking bituminous coal, using separately lump, nut, and pea coal. He gives as results of his experiments that, beside the direct saving in substituting the cheaper bituminous coal for the coke, jackets, slag and lead appeared hotter, the tuyeres brighter, and the crucible kept open better. The slag assays ran lower than with the usual coke and charcoal mixture, and the separation of slag and matte was good. On the feed-floor the charges settled more

* "Transactions of American Institute of Mining Engineers," xxv., p. 545.
† "*Ibid.*," xx., p. 165.

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evenly, as fewer zinc accretions were formed, and the top was cooler than usual; while the volume of smoke was larger, there was no greater loss in the fluedust on account of the charge being cooler. In his furnace, 36 by 78 in. at the tuyeres, and 12 ft. from the tuyeres to the charging-door, he used coke and charcoal in the proportion of 3 : 1, and was able to have 27% of bituminous coal in his fuel charge. He expected to reach 33%, and thinks that with a higher furnace one-half of his fuel might be bituminous coal.

The example of Neill in using mixtures of coke and bituminous coal, instead of coke or coke and charcoal, has been followed by others and proved successful. Austin* gives a summary of 18 months' experience in the use of mixed fuels—coke, charcoal, and bituminous coal—in the blast furnace (having a 12-ft. working height) of the Germania Lead Works, Salt Lake City, Utah. The subjoined table contains the leading data:

Blast Furnace Fuel.	Cost Per Ton.	Character.	Per Cent Ash.	Composition of Ash, Per Cent.				Calorific Power: Grams Lead Reduced by 1 Gram Fuel.
				SiO_2.	FeO.	CaO.	Cu.	
Coke from Castle Gate, Utah.....	$7.50	Friable	14.0	35.0	25.7	6.5	2.6	24.5
Coke from Crested Butte, Colo....	11.00	Fairly hard	11.3	32.2	30.4	6.6	2.8	26.5
Nut coal from Castle Gate, Utah..	3.00	Friable	8.4	42.0	29.8	10.0	.....	24.0
Charcoal.....	14.00	.....	....	.....	.....	.....	.....	.....

Blast Furnace Fuel.	Coke: Coal.			Coke: Coal.			Coke: Charcoal: Coal.		
	Proportions in Mixture.	Cost Per Ton Charge.	Per Cent of Mixture.	Proportions in Mixture.	Cost Per Ton Charge.	Per Cent of Mixture.	Proportions in Mixture.	Cost Per Ton Charge.	Per Cent of Mixture.
Coke from Castle Gate, Utah.....	14:2	$1.33	16	12:4 9:9	$1.04 0.94	16 18	9:2:6	$1.07	17
Coke from Crested Butte, Colo....	14:2	1.82	16	12:5 9:9	1.47 1.26	17 18	.....	.....	.....
Nut coal from Castle Gate, Utah..	.....	.....	.....	.....	.....	.....	.....	.....	.....
Charcoal....	.....	.....	.....	.....	.....	.....	.....	.....	.....

The conclusions drawn by Austin are, that in replacing charcoal and some of the coke by bituminous coal, the amount of fuel

* *Engineering and Mining Journal*, Dec. 15, 1894.

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required to the ton of charge has to be increased by 1 or 2%; that the effect is similar to that of charcoal in promoting the running of the furnace, but that an excess of bituminous coal is liable to choke up the charge and can be corrected by substituting charcoal; that bituminous coal appears to work unfavorably with coppery and arsenical ores, the crucible filling up slowly; that as regards manipulation the feeding of bituminous coal must be stopped whenever yellow smoke issues from the surface of the charge; that over-fire is to be avoided, as it ignites the explosive gas-mixture on its passage into the downcomer, and that lastly the blast may never be taken off entirely (at least one blower must be kept going), as the same mixture coming in contact with air in the furnace will be ignited by the hot charge. The replacing of charcoal and coke by bituminous coal is recommended, if the latter is considerably cheaper, and if copper and arsenic are present in small enough quantities to permit its use.

Coke and Anthracite.—Dwight* publishes some experiments made by Ropp in substituting anthracite of goose-egg size for part of the coke, no charcoal being used. The furnace was 36 by 80 ft. at the tuyere level and 9 ft. from there to the feed-floor. The result was that the smelting power of the furnace was reduced as the proportion of anthracite was increased, *e.g.*, one-third with anthracite as 60% of the fuel; otherwise the furnace remained in good condition. The top kept cool and the crucible open; there was a good reduction shown by a clean slag (0.4 to 0.8% lead), a matte low in lead (8% lead, 4% copper), and a speise having a coarsely crystalline structure. Finally, less zinc accretions were formed than when coke alone was used, the charge containing 7.5% zinc.

Gaseous Fuel.—This has been used in a single instance. At the works of the Pennsylvania Lead Company† Blake introduced natural gas with the blast by inserting a gas-pipe through the tuyere-pipe. The amount of natural gas was regulated by stop-cocks and the blast-pressure increased so as to supply sufficient air for the combustion of the gas. Thirty per cent. of coke was successfully saved in this way. By replacing 60%, the top of the furnace became too hot. That solid fuel cannot be entirely replaced by gaseous is clear from the reactions going on in a blast

* "Transactions of American Institute of Mining Engineers," xx., p. 169.
† *Ibid.*, xv., p. 661.

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furnace that require solid carbon. For pecuniary reasons it is improbable that any artificial gaseous fuel will ever be used in the blast furnace.

Weight of Fuel.—The weight of the fuel required in a lead furnace is generally expressed in terms of percentage of the total weight of the charge (ore, flux, and slag), sometimes by saying that one part of fuel bears so and so many times its weight of charge; the latter is more common in copper smelting. Misunderstanding occasionally arises from deducting the pounds of lead contained in the charge and referring the percentage of fuel only to the slag and matte material. Often the percentage of fuel used refers to the weight of the charge excluding the slag that is added, for the reason that an addition of slag does not within ordinary limits call for extra fuel because it facilitates the smelting. The amount of fuel required varies with its character, with the fusibility of the charge, with the time of year, and the altitude at which the smelting is carried on, and the pressure of the blast.

As regards the character of the fuel: coke that is rich in ash is not only an inferior fuel in proportion to the smaller amount of carbon it contains, but a considerable quantity of this carbon is consumed to melt the ash and the fluxes necessary to slag it. For this reason a smaller amount of charcoal than of coke would seem to be required. The exact opposite is, however, the case. This is mainly because the charcoal crumbles and is crushed in its descent in the furnace. There are, however, two other reasons: one is that the charcoal being porous burns more to carbon monoxide at the tuyeres than the denser coke and thus produces less heat, which has to be made up by additional fuel; the other is that charcoal in its descent through the furnace is oxidized more quickly by the carbon dioxide of the ascending gases than is the coke. Thus Bell* found that in conducting carbon dioxide through three strongly-heated porcelain pipes filled with dense coke, porous coke, and charcoal, the gas on leaving the pipes contained 5.4, 30.2, and 68.4 volumes of carbon monoxide. Akerman,† heating two tubes filled with coke and charcoal to 900° C., and passing carbon dioxide through them at a speed of 33 mm. per second, found that the carbon dioxide on leaving

* *Journal of the Iron and Steel Institute.* 1872, i., p. 74.
† *Oesterreichisches Jahrbuch*, 1889, p. 127.

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them contained 2.5 and 13.3 volumes of carbon monoxide, the smaller amounts of carbon monoxide being due to the lower temperature.

The richer an ore is in lead and the more fusible the rest of the charge, the less fuel will be needed. For instance, an ore containing zinc requires more fuel than one that is free from it; a calcareous slag requires more fuel than one that is furruginous; a coarse and open charge requires less fuel than one that is fine and dense.

In summer less fuel is generally required than in winter, not so much owing to the higher temperature, which may be counted as balanced by the larger percentage of moisture in the air, as to the more rapid evaporation of the water contained in ore, flux, and fuel. The difference may be as much as 5%.

The altitude at which an ore is smelted makes a great difference in the amount of fuel required. Hahn,* for instance, in 1881, gave the figures of 14 and 17% in Salt Lake City (4,000 ft. above the level of the sea) as against 22 and 24% in Leadville (10,000 ft.); the lower figure refers to the summer, the higher to the winter season. The only explanation that is at least in part satisfactory is the one given by Headden at a meeting of the Colorado Scientific Society, where the matter was informally discussed. A cubic foot of air entering the blast furnace under a certain pressure will expand more at a high elevation, where the air is rarified, than at sea-level; consequently more heat will be consumed, and this has to be made up by an extra amount of fuel. In the same way more power and consequently more fuel is required at a high elevation to obtain this cubic foot of compressed air.

The pressure of the blast has a great influence on the fuel consumption. With the pressure of ¾ and 1 lb., common three or four years ago, the amount of coke needed at an elevation of 5,000 ft. was about 15%, and at 10,000 ft. 18% and as much as 22%, if a mixture of coke and charcoal was used. This has fallen with a pressure of from 2 to 2½ lb. to 12 and 15% respectively. The reason for this saving is partly because of the smaller amount of carbonate ore treated, but mainly on account of the more even and perfect combustion of the coke at the region of the tuyeres, because the blast comes in better contact with the

* "Mineral Resources of the United States," 1882, p. 339.

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fuel. If, however, the pressure be strong enough for the air to penetrate much into the coke the carbon will burn only to carbon monoxide, with the consequent reduction of temperature; the porosity of the coke must therefore be a gauge for the blast.

With charcoal, where high-pressure blast is not permissible, the older figures hold good to-day, as much as 26 and 28% being used, according to Hahn.*

§ 71. CHEMISTRY OF THE BLAST FURNACE.—The reactions of a lead blast furnace have never been fully studied. Guyard† has a very interesting theoretical discussion of the subject in his paper, "Argentiferous Lead Smelting at Leadville," but measurements of temperatures and analyses of gases are wanting. Schertel‡ has measured the melting temperatures of the slags and analyzed the gases of the Freiberg blast furnaces, where slagged ores are smelted. With the aid of these two writers, whose work will be used to supplement the more general outline, the sequence of processes in the lead blast furnace can be approximately given.

The main reactions that take place in the blast furnace are reduction and precipitation; the latter has already been discussed (§§ 7 and 54). The principal reducing agents§ are carbon and carbon monoxide.

(*a*) Carbon acts on metallic oxides if its own affinity for oxygen is greater than that of the metal combined with the oxygen, and this affinity stands in direct proportion to the height of the temperature. The product of carbon and oxygen will be carbon dioxide, if the carbon acts on readily reducible oxides (lead, copper). With oxides not easily reduced (iron, manganese) carbon monoxide is the principal product.

$$2PbO+C=Pb_2+CO_2,$$
$$Fe_xO_{y+1}+C=Fe\ O_y+CO,$$
$$Fe_xO_y+yC=Fe_x+yCO.$$

(*b*) Carbon monoxide is formed in the blast furnace: first, by the direct combustion of carbon at a high temperature;

$$C+O=CO,$$
$$yC+Fe_xO_y=yCO+xFe;$$

* "Mineral Resources of the United States," 1882, p. 339.

† Emmons, "Geology and Mining Industry of Leadville," p. 731.

‡ *Freiberger Jahrbuch*, 1880, p. 37; *Wagner's Jahresberichte*, 1880, p. 188.

§ Balling, "Compendium der Metallurgischen Chemie," p. 63; Wright in *Colliery Guardian*, Jan. 3, 1890, or Roberts-Austen, "An Introduction to Metallurgy," London—Philadelphia, 1894, p. 243.

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secondly, by carbon dioxide being split into carbon monoxide and oxygen,

$$CO_2=CO+O,$$
$$CO_2+Fe_xO_{y-1}=CO+Fe_xO_y,$$
$$yCO_2+xFe=yCO+Fe_xO_y;$$

thirdly, by carbon dioxide combining with carbon,

$$CO_2+C=2CO.$$

The reducing power of carbon monoxide is on the whole favored by a high temperature. Its product of combustion is carbon dioxide.

$$PbO+CO=Pb+CO_2,$$
$$Fe_xO+CO=Fe_xO_{y-1}+CO_2,$$
$$Fe_xO+yCO=xFe+yCO_2.$$

It has always been accepted that carbon dioxide begins to be decomposed into carbon monoxide and oxygen at 1200° C., and cannot exist at 2000° C. If this be so, it follows that the reducing power of carbon monoxide is diminished as the temperature rises above 1200° C., and the same mixture of carbon monoxide and carbon dioxide may be a reducing agent at a low and an oxidizing agent at a high temperature. According to Blass,* the dissociation, however, begins only at from 1800 to 2000° C. and is only 4% at 2460° C. Langen and Meyer† state that carbon dioxide remains undecomposed at 1690° C. As these temperatures are not reached in a lead blast furnace the decomposition of carbon dioxide by heat alone does not concern the lead smelter.

(c) Other sources of carbon dioxide in furnace gases are:

First, the direct combustion of carbon at a low temperature,

$$C+O_2=CO_2.$$

Second, the splitting of carbon monoxide into carbon and carbon dioxide at a temperature of from 400 to 450° C. when in contact with iron oxide.

$$2CO=CO_2+C.$$

Third, the decomposition of carbonates: lead carbonate, 170 to 200° C.; magnesium carbonate, 650° C.; ferrous carbonate, about 800° C.; calcium carbonate, 812° C.‡

* *Stahl und Eisen*, 1892, p. 904.
† *Ibid.*
‡ Le Chatelier, *Thonindustrie Zeitung*, 1886, p. 429.

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Fourth, its entrance into the blast furnace in small quantities with the blast: 0.04% by volume.

The action of sulphur as a reducing agent has already been discussed (§ 9).

As has been seen, the carbon compounds in furnace gases are mainly carbon dioxide and carbon monoxide. The oxygen of the air, on entering the furnace through the tuyeres, meets incandescent fuel, which is converted into carbon dioxide or carbon monoxide according to the prevailing temperature. This depends on the melting point of the slag.

Freiberg lead slags melt, according to Schertel, at 1030° C.; other temperatures* given for lead slags are 1220° C. and 1273° C.; Guyard assumes 1200° C.; Ilest finds 1034° C.; Goetz‡ says the melting point of the slag: SiO_2 35.5, FeO 22.8, MnO 4.,5 CaO 21.7, MgO 4.6, Pb 0.7, Ag 1.0 oz., ranges from 1100 to 1200° C. and its formation temperature is 1300° C., the specific heat being 0.18.

In the products of combustion carbon dioxide will therefore strongly prevail over carbon monoxide. In ascending, some of the carbon dioxide will be reduced by carbon to carbon monoxide; to this will be added that which is produced by carbon acting on oxides of iron. These reactions decrease very quickly as the ascending gases expand at the boshes and come in contact with the cooler parts of the charge; here the percentage of carbon dioxide again increases quickly through the reducing action of carbon and carbon monoxide on readily reducible oxides. Thus the percentage of carbon dioxide in furnace gases increases, with one slight interruption, with their distance from the tuyeres, and that of carbon monoxide correspondingly decreases. The amount of carbon dioxide will be further increased by the decomposition of carbonates in the charge.

Of the 19 analyses of gases from Freiberg blast furnaces made by Schertel,§ a few, selected by Kerl,‖ may be quoted. The gases were taken from the throat of the furnaces. I. represents gases from smelting thoroughly slag-roasted ore to which oxidized

* Balling, "Metallhüttenkunde," p. 618.
† *School of Mines Quarterly*, xvii., p. 20.
‡ Private communication by A. S. Dwight, August, 1896.
§ *Freiberger Jahrbuch*, 1880, p. 87; *Wagner's Jahresberichte*, 1880, p. 188.
‖ *Berg- und Hüttenmännische Zeitung*, 1880, p. 85.

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lead from the refining works and roasted matte have been added; II., gases from resmelting the ore slag with roasted matte.

Volumes.		I. Ore Smelting.			II. Slag Smelting.		
In 100 Volumes of Furnace Gas.	N	72.72	75.30	75.20	75.5	76.0	74.8
	CO_2	16.26	17.80	17.20	16.6	17.4	18.5
	CO	10.06	5.20	5.40	5.9	4.3	3.5
	CH_4	0.36	0.10	0.70	2.0	1.8	0.3
	H	0.59	1.60	1.50	.....	0.5	2.6
For Every 100 Volumes of N.	CO_2	22.3	23.7	22.9	21.9	22.9	24.7
	CO	13.8	6.9	7.2	7.8	5.7	4.7
	C	18.1	15.3	15.0	14.8	14.2	14.7
	O	29.3	27.1	26.5	25.6	25.6	27.0
	Excess of O	+2.8	+0.6	.....	−0.9	−0.9	+0.5

The presence of hydrogen is explained by Schertel through the decomposition of marsh gas. In none of the analyses is sulphur dioxide taken account of. The largest amount obtained from a number of determinations was 0.15%. The ore being slag-roasted, very little, if any, sulphur dioxide will be set free in the upper parts of the furnace; most of the little liberated in the lower parts is reduced to sulphur when passing through the incandescent fuel and forms a sulphide.

Schertel concludes from the analyses, in which the relation of carbon dioxide to carbon monoxide by weight is as 4 : 1, that at the tuyeres most of the carbon is converted into carbon dioxide. He says that atmospheric air contains, for every 100 volumes of nitrogen, 26.5 of oxygen. If the oxygen of 126.5 volumes of air is completely converted to carbon dioxide, there will result 126.5 volumes of gas, containing 100 volumes nitrogen and 26.5 of carbon dioxide, equal to 13.25 volumes of gaseous carbon. If on the other hand the oxygen is converted completely into carbon monoxide, the resulting gas will consist of 100 volumes nitrogen and 53 volumes of carbon monoxide, equal to 26.5 volumes of gaseous carbon. From the tables it is evident that the proportion of 100 volumes nitrogen to 13.25 volumes gaseous carbon corresponds more closely to the figures of gaseous carbon in the analyses than would 100 volumes nitrogen to 26.5 volumes gaseous carbon. Consequently the carbon burns before the tuyeres rather to carbon dioxide than to carbon monoxide.

The slags produced in the furnaces from which Schertel made

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the gas analyses contain 4.75% lime and 0.54% magnesia. This would not increase the amount of carbon dioxide to any extent.

A few analyses of gas from Colorado smelting works are subjoined. They are from blast furnaces run with mixed sulphide and oxide ores on slags containing high percentages of lime, a blast of high pressure being used. It will be noted that the gases from Works A contain considerable free oxygen while those from Works B contain very little. The small percentage of carbon monoxide is probably due to the high-pressure blast.

	Works A.				Works B.			
N.....	74.1	69.4	73.0	68.6	77.6	78.9	79.6	75.5
CO_2.....	15.2	17.6	17.4	21.6	18.8	19.6	16.6	18.2
CO.....	9.7	8.0	5.4	10.8	3.0	1.5	3.6	5.9
O.....	1.0	5.0	4.2	0.4	0.6	0.0	0.2	0.4

The charge in passing from the throat of the furnace undergoes the following changes: It first loses its hygroscopic water, then that which is chemically combined. During the stages of incipient redness (525° C.) and dull redness (700° C.), oxide of lead will react on sulphides of lead, iron, and silver, unless already combined with silica or reduced to metal. Lead sulphate may also act on sulphide; its reduction begins as well as that of porous ferric oxide (oxide ore, roasted matte). Carbonate of lead is decomposed, dolomite loses part of its carbon dioxide, and the reduction of carbon dioxide by carbon begins.

As the charge descends through incipient cherry-redness (800° C.) and cherry-redness (900° C.) the reactions begun in the upper zone are about completed, except the reduction of carbon dioxide, and that of porous iron oxides by carbon monoxide. The action of carbon on iron oxides now begins. The lead set free reacts on lead sulphate, arseniate, and antimoniate, if these have not been reduced to sulphide, arsenide, and antimonide by means of carbon and carbon monoxide. The lead also reacts on silver chloride and sulphide. The charge begins to become pasty; iron and lime carbonates are decomposed; any lead sulphate still existing is converted into silicate; the reduction of iron oxide by carbon monoxide decreases, that by carbon grows; sulphides and arsenides rich in lead form to a greater extent and are in part decomposed by metallic iron. Ascending vapors of metallic zinc are sulphurized and oxidized.

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Passing through the stages of a clear cherry-red (1000° C.) and a deep orange (1100° C.) to a clear orange (1200° C.), fusion is perfected. Ferrous oxide and lime combine with silica, setting free lead oxide, most of which is reduced by carbon to metal; what remains may oxidize metallic iron to magnetic oxide, sometimes found in matte and slag; metallic iron acts on sulphides and arsenides, setting free some lead, and forms the matte and speise, with prevailing iron, as tapped from the furnace. Lime and baryta may also act as desulphurizing agents, carrying some calcium and barium sulphide into the slag; some zinc oxide will be reduced and vapors ascend in the furnace.

The melted masses separate according to their specific gravities, the lead collecting in the crucible and passing off through the lead-well; then come speise, matte, and slag, which are either tapped at intervals into the slag-pots, where they separate again in horizontal layers, or speise and matte alone are tapped, while clean slag overflows continuously.

§ 72. Calculation of Charge.*

Introductory Remarks.—In calculating a charge for the lead blast furnace, the typical slag best corresponding to the character of the ore is selected, and the necessary amount of fluxes and fuel then determined. Other considerations are the amount of lead that the charge will contain, the richness of the bullion to be produced, and the quantity of speise, matte, and slag that will ensue from the charge. A complete calculation will give full information on all these points.

To begin with the necessary amount of lead, which is expressed in percentage having reference to the sum of ore and fluxes, charges with as little as 6.5% of lead have been run successfully; the highest, with from 25 or 30%, is hardly ever reached. With pure ores, containing little or no zinc, arsenic, or antimony, and not much sulphur, it is safe to go as low as 8%; if these impurities are present to any extent the charge should contain not less than 12%. Ordinarily the lead in charges ranges from 12 to 15%. It is to be noted that more lead is lost by volatilization with a charge low in lead than with one that is high; the loss in

* Murray, *Engineering and Mining Journal*, Aug. 13, 1887, and March 5, 1892; Newhouse, *School of Mines Quarterly*, ix., p. 373; Furman, *Ibid.*, xiv., p. 134, also "Manual of Practical Assaying," New York, 1893, p. 337; Saint Dizier, *Colorado State School of Mines Scientific Quarterly*, vol. ii., No. 1, p. 50.

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silver depends mainly on the loss of lead and on the richness of the base bullion.

Certain permissible assumptions have entered into the subjoined calculation in order to simplify the work. For instance, all the zinc is figured as entering the slag, while in reality perhaps only 80% does so, the rest being volatilized or entering the base bullion, matte, and fluedust. No account is taken of any loss of lead, silver (gold), sulphur, arsenic, etc. All the lead and silver (gold) is assumed to be collected in the base bullion, while as a matter of fact some of it is lost and some goes into the by-products, especially the matte and fluedust. All the sulphur and arsenic of the ore are accounted for in the matte and the speise, while, especially with mixed sulphide and carbonate ores, a very large percentage of both passes off with the furnace gases and some arsenic enters the base bullion. The actual deviations from the assumed figures vary indefinitely with the character of the ores and slags and the working of the furnace.

For calculating a charge, a carbonate ore containing some galena may serve as an example. Its composition is:

SiO_2.	FeO.	MnO.	CaO.	MgO.	BaO.	ZnO.	Al_2O_3.	S.	As.	Pb.	Cu.	Ag_2Oz.	Au,Oz.
32.6	14.8	4.3	2.2	5.26	1.5	2.4	2.5	4.4	0.5	20.7	2.9	50.5	Trace.

The typical slag shall be:

$$30SiO_2—40FeO—20CaO.$$

The charge shall weigh 1,000 lb. and contain 10% of slag; the fuel—coke—shall be 15% of the charge. The analysis of the iron ore shows:

SiO_2.	FeO.	MnO.	CaO.
4.3	72.4	1.7	3.1;

that of the dolomitic limestone,

SiO_2.	FeO.	CaO.	MgO.
2.7.	4.5	37.3	11.9.

The coke contains 10% of ash, consisting of

SiO_2.	FeO.	CaO.	MgO.	Al_2O_3.
40.3	26.5	6.9	2.4	20.4

Before beginning the calculation it is necessary to bring the

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different slag-forming components of ore, fluxes, and fuel under the three main heads of silica, ferrous oxide, and lime.

The atomic weights of iron and manganese being very nearly the same, 56 and 55, the two oxides are simply added. Both ferrous oxide and metallic iron have to be considered in the calculations:

$$FeO \times \frac{7}{9} = Fe; \quad Fe \times \frac{9}{7} = FeO.$$

It will also be necessary to express the equivalents of one component (ferrous oxide) in terms of the other two (silica and lime).

$$\text{Let } SiO_2 = c,\ FeO = a,\ CaO = b, \text{ and } a + b + c = 90.$$

$$FeO : SiO_2 : : a : c. \qquad FeO : CaO : : a : b,$$

$$FeO = \frac{a}{c} SiO_2. \qquad FeO = \frac{a}{b} CaO.$$

Magnesia and baryta are generally classed under the head of lime, although some persons object to it. For instance:

$$CaO : MgO : : 56 : 40.$$
$$CaO = MgO \times 1.4.$$

In the same way $CaO = BaO \times 0.4$.

Some metallurgists bring also zinc oxide under the head of lime:

$$CaO = ZnO \times 0.7,$$

thus cutting down the lime of a slag with the increase of zinc oxide.

The analyses of ore, fluxes, and coke-ash, changed as indicated, are:

	SiO_2.	FeO.	CaO.	ZnO.	Al_2O_3.	S.	As.	Pb.	Cu.	Ag. Oz.	Au, Oz.
Lead ore.....	32.6	19.1	10.16	2.4	2.5	4.4	0.5	20.7	2.9	50.5	Trace.
Iron ore.....	4.3	74.1	3.10	.....	.....	.....	.....	.....	.....	.....	.....
Limestone	2.7	4.5	53.96	.....	.....	.....	.....	.....	.....	.....	.....
Coke-ash.....	40.3	26.5	10.26	.....	20.4	.....	.....	.....	.....	.....	.....

In figuring the charge five calculations have to be made to find:

1. The amount of available ferrous oxide and metallic iron;

2. The amount of metallic iron required by the arsenic to form Fe_5As;

3. The amount of metallic iron required to combine to FeS with the sulphur not taken up by the copper as Cu_2S;

4. The amount of flux required for the 15 lb. of ash in the 150 lb. of coke;

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5. The amount of flux required to slag the silica of the lead ore.

1. Available ferrous oxide and metallic iron in the iron ore. In the slag, 30 SiO_2 require 40 FeO. In 100 lb. iron ore there are 4.3 lb. SiO_2. These require

$$SiO_2 : FeO : : 30 : 40 : : 4.3 : x;$$
$$x=5.7 \text{ FeO}.$$

The iron ore contains 74.1% FeO; deducting 5.7 gives 68.4 available FeO, or $\frac{3}{4}$ FeO=53.2 available Fe.

2. Arsenic and iron. 100 lb. lead ore contain 0.5 lb. As.

$$As : Fe_5 : : 75 : 280 : : 0.5 : x;$$
x=1.86 Fe. How much iron ore is required?
Iron ore: available Fe : : 100 : 53.2 : : y : 1.86;
y=3.5 lb. iron ore.

3. Copper, sulphur, iron. 100 lb. of lead ore contain 2.9 lb. Cu.

$$Cu_2 : S : : 126.8 : 32 : : 2.9 : x;$$
$$x=0.73 \text{ S}.$$

Of the 4.4 lb. S contained in the 100 lb. of lead ore, 0.73 are required for the Cu; the difference, 3.67, must be combined with Fe.

$$S : Fe : : 32 : 56 : : 3.67 : y;$$
y=6.42 Fe, which corresponds to 12 lb. of iron ore, viz:
Iron ore : available Fe : : 100 : 53.2 : : z : 6.42;
z=12.

For the arsenic and sulphur of 100 lb. of lead ore 15 lb. of iron ore are required. These have: 0.66 lb. SiO_2, 0.48 lb. CaO, and 0.88 lb. FeO. Only the non-available FeO enters the slag according to 30 SiO_2 : 40 FeO; the rest, *i.e.*, the available FeO, combining as Fe with the As and S to form speise and matte. The 0.66 lb. SiO_2 require

$$SiO_2 : CaO : : 30 : 20 : : 0.66 : x;$$

x=0.44 CaO, which is balanced by the 0.48 CaO already present. If this were not the case, the 0.44 would have to be supplied by limestone:

$$\text{Limestone} : CaO : : 100 : 53.96 : : y : 0.44.$$

4. Coke-ash, 100 lb.

Murray's Method.—The method used resembles very closely

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the one given by Murray* and elaborated by Newhouse.† The analyses show:

Desired Amount.	Material.	SiO_2.	FeO.	CaO.
100	Coke-ash	40.3	26.5	10.26
x	Iron ore	4.3	74.1	3.10
y	Limestone	2.7	4.5	53.96

Starting again with 100 lb. of coke-ash, the necessary quantities of iron ore (x) and limestone (y) can be found by expressing the amounts of FeO first in terms of CaO, then in terms of SiO_2 (see above), and finally by putting these quantities equal to each other, when x and y can be easily calculated.

$$FeO=\frac{a}{b}CaO,$$

$$26.5+0.741x+0.045y=\frac{40}{2.}(10.26+0.031x+0.539y),$$

$$x=1.521y,-8.80.$$

$$FeO=\frac{a}{c}SiO_2,$$

$$26.5+0.741x+0.045y=\frac{40}{30}(40.3+0.43x+0.027y),$$

$$x=39.6-0.013y,$$

$$1.521y-8.80=39.6-0.001y,$$

$$y=31 \text{ lb. limestone},$$

$$x=39 \text{ lb. iron ore}.$$

5. Lead ore, 100 lb. The analyses give:

Desired Amount.	Material.	SiO_2.	FeO.	CaO.
100	Lead ore	32.6	19.1	10.16
x	Iron ore	4.3	74.1	3.10
y	Limestone	2.7	4.5	53.90

$$FeO=\frac{a}{b}CaO,$$

$$19.1+0.741x+0.045y=\frac{40}{20}(10.16+0.031x+0.539y),$$

$$x=1.8+1.52y.$$

$$FeO=\frac{a}{c}SiO_2,$$

$$19.1+0.741x+0.045y=\frac{40}{30}(32.6+0.043x+0.027y),$$

$$x=35.46-0.013y.$$

* *Engineering and Mining Journal*, Aug. 13, 1887; March 5, 1892.
† *School of Mines Quarterly*, ix., p. 373.

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$$1.8+1.52y=35.46\ 0.013y.$$
$$y=22 \text{ lb. limestone,}$$
$$x=35 \text{ lb. iron ore.}$$

6. Summing up. In summing up there are: 15 lb. coke-ash requiring 6 lb. iron ore and 4 lb. limestone; then the charge contains 100 lb. slag—a total of 125 lb.

The difference from 1,000 lb. = 875 lb. is to be made up by ore and fluxes. Now 100 lb. of lead ore require to slag the SiO_2, 35 lb. iron ore and 22 lb. limestone; to combine with As and S, 15 lb. iron ore are necessary; a total of 172 lb. $172x=875$; $x=5.088$. This gives as charge in round figures: Coke-ash, 15 lb. (=150 lb. of coke); slag, 100 lb.; lead ore, 510 lb.; iron ore for SiO_2, 185 lb.; iron ore for As and S, 75 lb.; limestone, 115 lb.; or 1,000 lb.

Figuring the pounds of each component of the charge and adding like to like must give the slag.

By $216.57x=30$, the coefficient is obtained with which the totals of SiO_2, FeO, and CaO have to be multiplied to obtain the desired figures: 30, 40, 20. The table shows that the calculation is correct.

Material.		SiO_2.		FeO.		CaO.		ZnO.	
Name.	Dry Weight Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.
Coke-ash	15	40.3	6.04	26.5	3.97	10.26	1.54	.....	.....
Slag	100	30.0	30.00	40.0	40.00	20.00	20.00	.....	.....
Lead ore	510	32.6	166.26	19.1	97.41	10.16	51.82	2.4	12.2
Iron ore (SiO_2)	185	4.3	7.95	74.1	137.08	3.10	5.73	.....	.....
Iron ore (As, S)	75	4.3	3.23	74.1	4.29	3.10	2.32	.....	.....
Limestone	115	2.7	3.10	4.5	5.17	53.90	61.99	.....	.....
Totals	1,000	.....	216.57	.....	287.92	.....	143.40	.....	12.2
Coefficient	0.1385	}	29.99		39.87		19.86		1.69

Material.		Al_2O_3.		Ag.		Pb.		As.		Cu.		S.	
Name.	Dry Weight Lb.	Per Ct.	Lb.	Oz. Per Ton	Oz.	Per Cent.	Lb.	Per Ct.	Lb.	Per Ct.	Lb.	Per Ct.	Lb.
Coke-ash	15	20.4	3.06	....	....	.....	.....	....	....	....	....	....	....
Slag	100	....	.....	....	....	.....	.....	....	....	....	....	....	....
Lead ore	510	2.5	12.75	50.5	12.8	20.7	105.6	0.5	2.5	2.9	14.8	4.4	22.4
Iron ore (SiO_2)	185	....	.....	....	....	.....	.....	....	....	....	....	....	....
Iron ore (As, S)	75	....	.....	....	....	.....	.....	....	....	....	....	....	....
Limestone	115	....	.....	....	....	.....	.....	....	....	....	....	....	....
Total	1,000	....	15.81	....	12.8	.....	105.6	....	2.5	....	14.8	....	22.4
Coefficient	0.1385		2.29 }	242.4 oz. bullion.				11.8 lb. speise.		69.9 lb. matte.			

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By adding the different components that go to form the slag, 676 lb. are obtained, and any change that is to be made in the slag must be calculated as having reference to this figure.

For every 12.8 oz. of silver there are 102.5 lb. of lead; the base bullion will therefore assay about 242.4 oz. silver to the ton.

There are 10% of lead in the charge.

In the charge are 2.5 lb. of arsenic, which, with 9.3 lb. of metallic iron, form about 12 lb. of speise.

The 14.8 lb. of copper, requiring 3.7 lb. of sulphur, will form 18.5 lb. of copper matte. Deducting the 3.7 lb. of sulphur from the total sulphur leaves 18.7 lb., which, with 32.7 lb. of metallic iron, give 51.4 lb. of iron matte; the total matte formed will be about 70 lb.

The table further shows that there are 10% of slag and 15% of fuel to the charge, thus giving all the necessary data.

It will be noticed that in summing up the results of the calculation the weight of the coke-ash (15 lb.) has been included, although not that of the coke. This is not usually done by lead and copper smelters who, contrary to the practice of the iron smelter, include only ores and fluxes and not the fuel. In this instance the coke-ash has been added for the purpose of bringing together in the table everything that influences the formation of the slag.

In making up the charge for the blast furnace, the moisture has still to be considered. If the lead ore contains, for instance, 5% of moisture, 535 lb. of moist ore will have to be used to correspond to 510 lb. of dry ore:

$$\text{Moist Ore: Dry Ore} = 100 : 95 :: x : 510;$$
$$x = 536.8.$$

The same is the case with fluxes and fuel.

It is to be noted that figuring a charge according to Murray's formula has one great advantage over the method next to be described, viz., that it shows in what proportions any three classes of silicious, ferruginous, and calcareous ores are best mixed so as to become self-fluxing.

Common Method.—In the common method the same ores, fluxes, and fuel, and the same slag as before, are taken as a basis.

The preliminary calculations, such as bringing the different components of ore, flux, and fuel under the heads of SiO_2, FeO,

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and CaO, are made in the same way. The total weight (1,000 lb.) that the charge is to have, and with it the percentage of fuel (15%) and slag (10%) to be added are fixed. The available FeO and Fe of the iron ore are determined as before.

Two calculations are now necessary to determine the amounts of iron ore and limestone required by the coke-ash and by the ore.

The analyses of the ash and the two fluxes, iron ore and limestone, are entered, as shown by the italic type in the table which follows. The 150 lb. of coke contain 15 lb. of ash; for these the totals of SiO_2, FeO, and CaO are figured and entered in the table.

There are 6.04 lb. of SiO_2; how much FeO is required?

$$SiO_2 : FeO :: 30 : 40 :: 6.04 : x;$$
$$x = 8.05 \text{ lb. FeO are necessary;}$$
$$3.97 \text{ lb. FeO are present.}$$

The difference, $y = 4.08$ lb. FeO have to be added.

To find the necessary iron ore:

Iron ore: Available FeO $:: 100 : 68.4 :: z : 4.08,$
$$z = 6 \text{ lb. iron ore.}$$

They are entered in the table; their total pounds of SiO_2, FeO, and CaO are figured and also entered.

To the previous sum of 6.04 lb. SiO_2 have been added, by the iron ore, 0.26 lb. SiO_2, making the total SiO_2, 6.30 lb., for which limestone has to be provided. How much CaO is required?

$$SiO_2 : CaO :: 30 : 20 :: 6.30 : u,$$
$$u = 4.20 \text{ lb. CaO are necessary;}$$
$$1.73 \text{ lb. CaO are present.}$$

The difference, $v = 2.47$ lb. CaO, have to be added.

To find the necessary limestone (leaving out the SiO_2 and FeO it contains):

$$\text{Limestone} : CaO :: 100 : 53.9 :: w : 2.47,$$
$$w = 4.5 \text{ lb. limestone,}$$

which are entered with the pounds of CaO they bring to the slag.

To see if the calculation is correct, the pounds of FeO and CaO are multiplied by a coefficient (4.76, from $6.30x = 30$, $x = 4.76$), which changes the pounds of SiO_2 to 30, the percentage of SiO_2 of the slag aimed at. The result will be 40 FeO and 20 CaO.

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The table gives 40.08 FeO and 19.75 CaO, showing the calculation to give sufficiently close results.

Material.		SiO_2.		FeO.		CaO.	
Name.	Dry Weight. Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.
Coke-ash	15.0	40.30	6.04	26.50	3.97	10.26	1.54
Iron ore	6.0	4.30	0.26	74.10	4.45	3.10	0.19
Limestone	4.5	2.70	.....	4.50	.....	53.90	2.42
Total	25.5	.....	6.30	.....	8.42	.....	4.15
Coefficient	4.76	.....	29.98	.....	40.08	.....	19.75

The weights of iron ore (6 lb.) and limestone (4½ lb.) are practically the same as those found by using Murray's method.

Deducting 125 lb. (the sum of coke-ash with its iron ore and limestone, and of slag) from the total weight of the charge of 1,000 lb. gives the same 875 lb. as before to be made up by the ore and its fluxes.

Now a table, like the one below, is laid out for the ore, the analytical data are entered, and the calculation is made on a basis of 100 lb.

1. The amounts of iron ore required (15 lb.) by the As and S are calculated as shown above, and the results are entered in the table.

2. 100 lb. of ore contain 32.6 lb. SiO_2, for which the necessary iron has to be provided:

$$SiO_2 : FeO :: 30 : 40 :: 32.6 : x;$$
$$x = 43.46 \text{ lb. FeO are necessary;}$$
$$19.10 \text{ lb. FeO are present.}$$

The difference, $y = 24.36$ lb. FeO have to be added.

To find the necessary iron ore:

$$\text{Iron ore} : \text{Available FeO} :: 100 : 68.4 :: z : 24.36;$$
$$z = 35 \text{ lb. iron ore.}$$

3. To the 32.60 lb. SiO_2 of the ore have been added from the two additions of iron ore $0.65 + 1.51 = 2.16$ lb. SiO_2, making the total of 34.76 lb. of SiO_2, for which lime has to be provided:

$$SiO_2 : CaO :: 30 : 20 :: 34.76 : u;$$
$$u = 23.17 \text{ lb. CaO are necessary;}$$
$$11.72 \text{ lb. CaO are present.}$$

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The difference, v=11.45 lb. CaO have to be added.

To find the necessary limestone (leaving out the SiO_2 and FeO it contains):

$$\text{Limestone} : \text{CaO} : : 100 : 53.90 : w : 11.45;$$
$$w = 21 \text{ lb. limestone,}$$

which are entered upon the table.

Adding the pound-columns of SiO_2, FeO, and CaO, and multiplying by 0.86, proves that the calculation is correct.

Material.		SiO_2.		FeO.		CaO.	
Name.	Dry Weight Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.
Ore	100	*33.60*	33.60	*19.10*	19.10	*10.16*	10.16
Iron ore for As and S	15	*4.30*	0.65	*74.10*	0.88	*3.10*	0.48
Iron ore for SiO_2	35	*4.30*	1.51	*74.10*	25.98	*3.10*	1.08
Limestone	21	*2.70*	.....	*4.50*	.....	*53.90*	11.32
Total	171	.....	34.76	.....	45.91	.....	23.04
Coefficient	0.86	.....	29.89	.....	39.58	.....	19.81

The figure found for iron ore (35 lb.) is the same as with Murray's formula; that for limestone is slightly lower (21 *vs.* 22 lb.), as the SiO_2 and FeO contained in the limestone are left out, making the available CaO 53.90, which is slightly too high.

If the items of the ore charge are now multiplied by 5.088 (as above) and those of the coke charge added, the sum of 1,000 lb., the entire charge, will again be obtained.

GENERAL SMELTING OPERATIONS.

§ 73. Blowing-in.—This consists of three parts: Warming the crucible, filling the furnace, and starting the smelting.

The warming of the crucible in a new furnace must be done slowly and with great care, so as to raise the temperature gradually; otherwise the quickly escaping moisture will crack the masonry and allow the lead in the crucible to percolate. The practice of metallurgists differs in the manner of warming the crucible and the amount of time given to it. The writer has, whenever it was possible, taken 48 hours to warm the crucible, and has been accustomed to proceed in the following way: the water is turned into the jackets so as to fill them and have just a little overflow; then the flue leading to the dust-chambers is

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closed, and the damper on top of the stationary stack raised, or the movable stack put in place, so that the gases may pass off into the open air.

With some furnaces only the iron plate for closing the furnace is lowered, and the joint covered with fine ore, the gases formed, when blowing-in, being allowed to pass off into the dust-chamber. This can only be done if the furnace is blown in with very little or no charcoal, which means having the crucible full of molten lead before the charge is introduced. If the furnace were filled to the top of the jackets with charcoal before adding the blowing-in charges, the gases, when the blast is let on, being rich in carbon monoxide, would be liable to ignite and damage the flue and dust-chamber.

A wood fire is made in the bottom of the furnace that will not reach halfway up the crucible. If it is kept going for a few hours, always replenishing the wood, ashes will have collected in the crucible. These, being bad conductors of heat, have to be removed in order that the burning wood may be in contact with the bottom. When raked out by means of a hoe from the breast of the furnace a new fire is kindled. After from three to four hours too many ashes will have accumulated in the furnace for the heat to have the desired effect, and the crucible is cleaned out again. While the crucible is being dried and warmed, the lead-well is filled with glowing charcoal, and the basin itself covered by a piece of sheet iron, so as to admit only a little air, thus preventing the charcoal from being burned quickly. Similarly the breast of the furnace is closed with loosely set bricks, by which the draught is checked and too quick combustion of fuel on the surface prevented. The heating is continued for 24 hours, when the outside of the crucible will feel warm to the touch. This shows that all the moisture is expelled and that the crucible can stand a high heat without endangering the brickwork.

The second warming has for its object the heating of the crucible to a good red-heat, and requires carbonized fuel and blast. A new fire is kindled and some charcoal added; the blower is started, and all the tuyere-bags are tied up or wound up, or the blast-gates are closed, except that of the tuyere nearest the breast to be used. This is connected with an iron pipe inserted into the crucible from the breast, and the blast allowed to play on the

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charcoal till it is well ablaze. A second layer of charcoal is added, and when it is fully aglow a third one, and so on till the crucible is filled to the jackets. The iron pipe is then inserted deep into the glowing coal, in order that the blast may reach the bottom of the crucible. Meanwhile the furnace-man works at intervals with an iron bar and a hoe, turning the coal over and moving it from front to back and *vice versa*, so as to get it all into a perfect glow. When this is accomplished, the pipe, of which often a small part has been melted off, is withdrawn, and the furnace is let alone for an hour or two. The pipe is again introduced and the charcoal burned down, the furnace-man stirring it to bring all parts into contact with the blast. The ashes are removed, and the crucible and well thoroughly cleaned. The heating is repeated from three to four times in 24 hours. Theoutside of the crucible will then have become too hot to be touched with the hand.

The second step is the filling of the furnace. This must be done with care, in order that a crust may not form later on the lead, when the blast has been started, as it is very troublesome to remove, and sometimes remains in the furnace during an entire run. To avoid a crust, it is essential to have a clean crucible entirely filled with red-hot lead. If half-filled with a mixture of charcoal ashes and small bits of charcoal, a dead layer will be formed between the lead and the slag which will soon harden to a crust and attract small obstacles that would otherwise be carried out by the slag. The proper mode of filling a blast furnace depends a good deal on the kind of fuel, and is to some extent individual with the metallurgist. After supplying the crucible with the necessary lead, which is either charged from the top or melted down from the breast of the furnace, the first charges will consist of an easy-smelting slag with much fuel and the necessary fluxes. Ore-charge gradually replaces the slag-charge, and the high percentage of fuel will be cut down until finally the normal charge is reached. It is better for a blowing-in slag to be glassy than crystalline, as it melts easier, and a slag consisting of 30 SiO_2, 40 FeO, and 20 CaO is probably the best.

Two methods of blowing-in may be mentioned, both requiring charcoal.

For the first, the thoroughly heated crucible is gradually filled with charcoal, the blast playing on it till it is all aglow; then the

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breast of the furnace is put in, the tuyere openings are closed, and the rest of the charcoal is added from the feed-floor, so as to reach about 1 ft. above the tuyeres. Then follows a bed of coke 1 ft. thick, which will reach about to the top of the jackets. Upon this are charged lead and coke, with some slag and the necessary fluxes. When all the lead has been given that is required to fill the crucible, the furnace is filled with alternate layers of fuel and of half ore-charge, half slag-charge, using the amount of fuel necessary for a full ore charge.

The amount of iron ore and limestone are calculated in the same way as in making up the regular blast-furnace charge.

The tuyere-holes in the jackets are now cleared, the lead-well is partly uncovered, the tuyere-pipes are put in place, and the blower is started slowly, the number of revolutions as well as the pressure of the blast being noted.

Too much emphasis cannot be laid on starting the blast slowly. If the blower is allowed to make too many revolutions per minute, the result is that the fuel burns quickly, the heat rises, and is not concentrated at the tuyere-level, and the gases rush upward and do not thoroughly warm the charge; the pressure of the blast at the same time lowers the level of the lead in the crucible, allowing an accumulation of charcoal ashes above. When finally slag and matte come down, they become partially chilled, because they sink too much below the level of the tuyeres, and thus the dreaded blowing-in crust is formed. For this reason the blower should always be started up slowly, and the pressure not allowed to be much over one ounce until slag appears before the tuyeres. The normal pressure of the blast is not reached until seven, eight or more hours after the blower is started.

When the blast has been turned on, a flame issues from the siphon-tap till the lower aperture is closed by the lead. Care must be taken that the passage of the well does not become choked with ashes, which are carried out by the flame in considerable quantities. The lead rises slowly as the bullion melts down in the furnace, and while often at first a little cool, it soon becomes hotter, and, when it reaches the top of the well, ought to be bright-red. Shortly after the blast has been let on, the flow of water into the jackets must be increased in order that it may not boil. Soon, however, when the slag charged begins to melt and come down, the jackets become coated and thus cooler.

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The large excess of water required just before is turned off, and less than the normal amount will be sufficient for an hour or two. On the feed floor the charges will at first sink quickly when the blast has been turned on, and it may be sometimes advisable to continue feeding alternately ore-charge and slag-charge until the furnace has been entirely filled with this mixture, and only then to begin with normal ore-charge. This is, however, exceptional. The smoke issuing at first from the top of the charge will be black, and gradually become lighter, and decrease in amount. The furnace-man watches through the peep-hole in the tuyere-pipe to see when it is time to tap the slag. This will be indicated by the appearance of a little blue flame of burning carbon monoxide gas.

The ashes of charcoal must now be removed from the furnace, the lead having lifted them up from the bottom to about the level of the tap-hole. They float on the lead, and are liable to prevent the necessary contact of lead and slag or lead and matte in the furnace. Generally the slag carries out the ashes, but sometimes this has to be assisted, which is done by entering with a rod through the tap-hole and loosening the ashes where they are inclined to accumulate and adhere, thus stirring them into the slag. It is often advisable after stirring with a rod to let the blast blow through the tap-hole, as it will blow out a good many ashes. If, nevertheless, a crust of ashes and slag should form on top of the lead, it is necessary to break it by thrusting bars through it and lifting it up while it is still soft. If it hardens, the furnace will have to be stopped, and the breast taken out before the crust can be removed (see § 77). The indications of a crust are that the lead in the well becomes dark and does not play freely with the blast. The first is easily seen; the second can be discovered by removing the plug or cap of the peep-hole in the tuyere-pipe just above the lead-well. If the lead plays freely, it will sink in the well; if not, it will remain immovable. By inserting a rod through the tap-hole the position of the crust can be felt.

If everything goes well, the furnace will have its full blast in about seven hours after starting, the slag its normal heat next day.

One of the main points in filling a furnace is to be quick about it; therefore as many men should be put to work as possible.

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A furnace 42 by 108 in. can be filled in from 30 to 40 minutes by two sets of feeders, of four each, helped by as many wheelers as may be required to bring the necessary materials to the furnace. If there is any delay in charging, the crucible may cool and a crust form. For this reason no weighing is done at the furnace, but the charge is measured by the number of scoopfuls and shovelfuls, the average weight of one having first been determined by weighing 10 scoopfuls of coke and 10 shovelfuls of iron ore and limestone.

With good charcoal this method of blowing-in is quite satisfactory; a crust very rarely forms. With charcoal of an inferior grade the second method is used. It is as follows: when the crucible has been well heated up and cleaned out as described, the breast is put in, the tuyeres in the jackets are closed, and the lead-well is covered with sheet iron, its lower side being coated with a clay lute and the upper one weighted with a couple of bars of lead. Charcoal is fed from the feed-floor until it reaches the top of the jackets, and covered by a bed of coke about 1 ft. in thickness. A man descends into the furnace and spreads out evenly the successive charges of bullion, coke, slag, iron ore, and limestone. Then follow the regular ore charges. When the furnace has been filled, which takes about two hours, the charcoal is kindled from the tuyeres and a gentle blast started. The work now proceeds in the usual way. When the lead, coming down in the furnace, has closed the lower opening of the well, the cover is removed. Whether this has occurred can be found by removing from time to time the iron rod which closes the tap-hole of the lead-well. The first slag is tapped about 25 minutes from the time that the lower opening of the well is closed by the lead, which shows how quickly the crucible fills.

The reason for the good results attained by this method may be explained in the following way: as the entire lower part of the furnace is closed during the filling, and the charcoal is not kindled until just before the blast is started, it cannot burn in the crucible and form the dreaded mixture of ash and fine coal. At the tuyeres much heat will be generated by the combustion of the charcoal. A slight effect may be communicated downward, but it proceeds principally upward. The first lead that melts and trickles down is very hot when at the tuyere-level, but cools somewhat below it. As more follows, the first becomes heated

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again, so that finally the bottom of the crucible is filled with liquid lead, although it is not yet very hot. The unburned charcoal is raised gradually toward the tuyeres, where it burns completely, leaving no mixture of ash and fine coal, and gives a very high temperature, which prevents the forming of a crust. The lead as it rises in the crucible becomes continually hotter from the addition of red-hot lead from above and its approach to the tuyere level, as well as by its contact with the hot slag.

Both of these methods require passably good charcoal, if not the best. If this cannot be had, it is replaced by dry split wood cut into pieces of suitable length. When wood is used instead of charcoal, it is burned down in such a way as to make fresh charcoal in the crucible. Thus, when the crucible is sufficiently heated and cleaned out, the last fire is made and the wood burned down quickly with the blast only so far as to be well charred, fresh wood being added and the burning down continued until the crucible is filled with charred sticks. The breast is put in place and a 2-ft. bed of coke given. Upon this come the regular blowing-in charges, the double amount of coke requiring an extra quantity of slag, iron ore, and limestone.

As it is difficult to get good charcoal, it is generally necessary to melt down the lead in the crucible before filling the furnace, as filling the crucible with coke is nearly sure to form a very disagreeable crust, which it is difficult to remove afterward from the breast. If neglected, it grows rapidly, and the result may be a crucible frozen up solid. The mode of procedure is to fill the well, after it and the crucible have been heated and cleaned, with glowing coals, and to pile them high up over it, so that it shall keep hot. Then as many bars of lead as the crucible will hold are introduced from the breast by sliding them in on a board. A fire is kindled on top of the lead; when that is melted down, the floating ashes are removed, a new fire is made, more lead melted down, and this repeated until the crucible is full of lead. The last ashes are raked out, enough of dry kindling that has been sorted out is charged from the front and then from the feed-floor to reach well above the jackets, and the breast is put in. On top of this comes a 2-ft. bed of coke, to which eight or ten bars of lead are added, in order that this, coming down hot, may help to heat the lead in the crucible. Then come the usual slag charges

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and slag-ore charges. The feeding of lead from the top is often continued until the lead in the crucible becomes red-hot.

Some metallurgists avoid charcoal altogether. The crucible is warmed with wood, the lead melted down with coke, all the clinkers are removed, a good coke fire is started on the lead, and the furnace filled with coke to the top of the jackets. Then a few slag charges are given and the furnace is filled with an easy ore-charge running high in lead. This is soon changed for the regular charge.

The writer has never blown in a furnace without using lead. Hahn* says that, when no lead can be had, he closes the syphon-tap where it enters the crucible with a clay plug, removing it when the lead begins to flow out with the slag. He adds that bottom crusts are unavoidable. When sufficient lead has been produced to fill the crucible the furnace is blown out and started up again in the normal way.

A method of blowing-in described by Henrich† refers rather to a furnace where metal is tapped from the bottom than to one having a lead-well. It is given here because it describes the blowing-in of a blast furnace used for concentrating matte, which is tapped from the bottom or discharges continuously into an external crucible. The crucible is warmed, filled with charcoal up to the level of the tuyeres, more fuel is given, and then blowing-in charges are fed, followed by ore charges. When the furnace is filled to from one-half to two-thirds the distance between the tuyeres and the charging floor, the front is closed. All the tuyeres excepting one or two (according to the size of the furnace) opposite the metal tap (2 in. wide) are closed, the pipes inserted, and the blower is started. The flame passes through the tap-hole, blows out ashes, and heats the bottom. The tap is kept open by inserting a rod until slag begins to flow, when it is closed with a clay plug. The other tuyere pipes are now introduced at short intervals one after the other. Once or twice slag is tapped from the metal tap; it is then allowed to accumulate in the crucible and tapping is begun at the slag tap. In this way the crucible becomes thoroughly heated.

The only instance coming under the writer's observation of slag being taken out from the lead-well was at a desilverizing

* "Mineral Resources of the United States," U. S. Geological Survey, 1882, p. 342.
† *Engineering and Mining Journal*, April 11, 1885.

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plant, when very rich bullion, obtained by smelting zinc-crusts in a circular 36-in. furnace (§ 119), had to be exchanged quickly for low-grade bullion. The furnace was fed with a number of slag charges to half fill it; then sufficient low-grade litharge and slag were added to fill the crucible again with lead. When the slag charges had about reached the tuyeres and much slag was being formed, the lead was all dipped out of the well and then the slag, until finally the lead of the litharge appeared and filled

TABLE OF BLOWING-IN CHARGES.

Charges, Number.	Coke, Scoops.	Charcoal, Scoops.	Iron Ore, Shovels.	Limestone, Shovels.	Bars of Bullion, Number.	Slag, Shovels.	Size of Furnace, Inches.	Crucible Filled From.
1	20	.....	3	2	6	.....		
3	15	.....	3	2	6	1		
5	10	.....	3	2	8	2	36 in diam.	Above.
5	10	.....	3	2	7	2		
2	10	.....	3	2	5	5		
5	10	.....	Half ore-charge.		3	6		
1	10	.....	Half ore-charge.		.....	6		
			Regular ore-charge.					
1	10	.....	.....	.....	6	.....		
3	8	.....	½	½	6	1		
4	7	.....	½	½	6	2		
4	6	1	½	½	6	2	36×84	Above.
2	5	2	½	½	5	2		
2	5	2	Half ore-charge.			2		
4	4	3	Half ore-charge.			2		
			Regular ore-charge.					
1	20	.....	3	2	6	.....		
3	15	.....	3	2	6	1		
10	15	.....	3	2	6	2	36×84	Above.
5	12	.....	3	2	4	3		
4	6	.....	Half ore-charge.			4		
			Regular ore-charge.					
1	25	.....	1	1	15	12	42×120	Above.
12	12	6	1	1	15	12		
			Regular ore-charge.					
1	25	.....	1	1	20	.....	42×120	Below.
15	9	.....	1	1	2	83		
			Regular ore-charge.					

the well, when slag was tapped again in the usual way. The rich bullion was very little diluted and could go to the cupelling furnaces.

The making up of blowing-in charges varies a great deal. The preceding table gives a few examples. A scoop of coke is assumed to weigh 12 lb., one of charcoal 4 lb., a shovel of iron ore 15 lb., of slag 12 lb., and of limestone 10 lb.

§ 74. Regular Work on the Charging Floor.—The work on the charging floor consists in bringing ore, flux, and fuel from

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the bins to the scales, weighing out the required amounts, dumping them, and feeding into the furnace.

Ores and fluxes are brought to the scales either in trucks or wheelbarrows; the former are filled through chutes from the bins, the latter by shoveling; coke and charcoal are nearly always brought in wheelbarrows.

Everything that goes in the furnace must be accurately weighed. No thorough work can be done without. An approximate weight can be obtained by taking 10 or 20 shovelfuls and reducing this weight to that of one, and then making up the

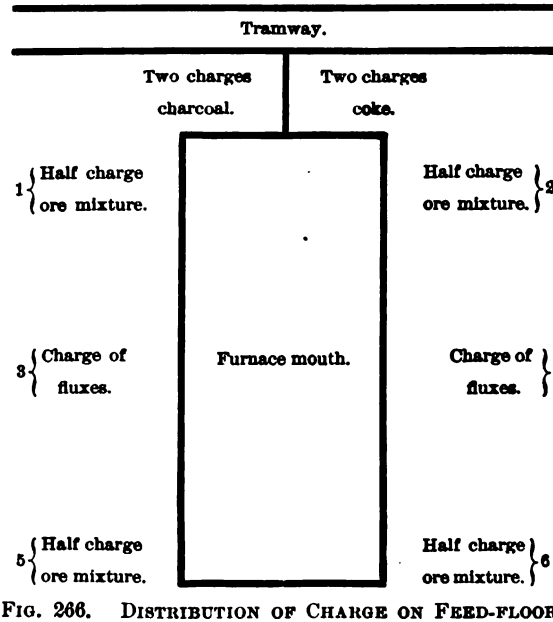


FIG. 266. DISTRIBUTION OF CHARGE ON FEED-FLOOR.

charge by measure. But shovelfuls differ according to the men who make them, and the same man will not always shovel a uniform amount—say, at the beginning and toward the end of his shift. With the shovel system the furnace is always liable to get out of order, and the metallurgist has no means of accurately determining the cause. As seen from § 73 the shovel system is used in ore smelting in blowing in a furnace, where the filling with slag charges must be done as quickly as possible; as soon as ore is substituted it must be regularly weighed. In refining works, where certain lots of lead by-products are treated in a

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blast furnace with slag alone, the shovel system is often found. This is permissible on account of the small amount of new slag that is being formed in the blast furnace, but even then weighing proves more satisfactory.

The scales in use have a capacity of from 1,000 to 1,500 lb. They are of two kinds: the single-beam scale, where the weights have to be shifted for every weight taken, and the multiple-beam, where by pulling a lever on the outside of the case the one of the four, six, or eight beams which bears the desired weight can be set free. By the multiple-beam errors are avoided, as the foreman sets the weights and locks the box surrounding them, the workman having only to pull the levers in their regular order.

In making up a charge and in distributing it in front of the feed-door the fuel is nearly always kept and fed separately, while ore, fluxes, and slag are sometimes mixed and sometimes kept separate. With small furnaces—say, 36 by 60 in.—ore, fluxes, and slag are usually dumped into one heap on the long side of the furnace, and the fuel has its place on the short side. With a large furnace (42 by 120 in.) ore and fluxes are sometimes distributed in separate heaps on the long sides of the furnace. Iles,* for instance, adopted the following disposition of the charge on the furnace floor (Fig. 266). The numbers denote the order in which the materials are supplied to the furnace. The feeder first spreads one-half of the charcoal pile on the preceding charge in the furnace, then one-half of the coke pile on the opposite side, and on top of this the ore mixture and the fluxes 1, 3, and 5. He then repeats the operation with the remaining charcoal and coke piles and covers them with the rest of the ore mixture and the fluxes 2, 4, and 6.

The mode of distributing the charge in the furnace deserves great attention, as by wrong feeding a good charge can be spoiled and a furnace be put out of order. Fuel and ore charge go into the furnace in alternating layers, beginning always with the former. The fuel must never be exposed to the air, but must be well covered by the ore charge. If, as with the mechanical feeding, there is danger of the fuel being broken up by the charge falling on top of it, it may be better to place the fuel last in the feeder and leave it somewhat exposed to the air in the furnace. In feeding, the fuel, especially charcoal, should, if possible, be

* *Engineering and Mining Journal.* March 24, 1883.

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kept near the center, and the charge distributed more toward the walls of the furnace, and, finally, the finer parts fed nearer the walls than the coarser ones. The reason for this is that in the descent of the charge the lighter fuel is liable to be pressed toward the walls of the furnace; further, as there is more friction between charge and furnace wall than between the parts of the charge itself, it will sink quicker in the middle while descending, and pack at the sides. If ascending gases are to pass evenly through the charge, their passage must be assisted in the center, and retarded at the sides. Lastly, with carbonate ores an intimate contact of readily reducible lead oxide with the fuel in the upper part of the furnace must also be avoided, as it would assist in reducing it to metallic lead, and thus increase the loss by volatilization. Guyard* thinks that this distribution of fuel toward the center, and of charge toward the furnace walls, favors the growth of accretions immediately above the jackets, and advises charging the fuel alternately toward the center and the sides. He says the effect of this would be that the accretions would begin to form higher up in the furnace and could be removed more easily. In feeding a charge that contains much fine ore (sand carbonates) special precautions are taken to prevent this from trickling through the charge, as it fills up the pores of the coke, and, arriving in a crude state at the smelting zone, chills the furnace and often fills the tuyeres. Glenn† describes what he calls a filter charge. It is not uncommon with lead furnaces, and consists in making the charge large, and in feeding gradually on the heavy coke bed first the coarse parts, then finer ones, and last of all the fines here and there, as the charge settles in the furnace. The finer the ore, the more uniform in size should the slag and fluxes be to make a good filter. Fine ore s are often fed where the fumes are strongest, thus helping to equalize them. The distribution of ore and fluxes in a charge may affect its fusibility. Henrich‡ gives some very interesting experiences on this point, and calls attention to the fact that the formation of the slag, while the charge is descending in the furnace, is a gradual process. Pieces of ore and flux containing the necessary constituents to form slag of a

* Emmons, "Geology and Mining Industry of Leadville," U. S. Geological Survey, monograph xii., p. 665.

† *Engineering and Mining Journal*, July 19, 1884.

‡ *Ibid.*, Dec. 27, 1890; June 6, 1891.

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low melting point may do this in the upper parts of the furnace if in close contact with one another. The liquid slag will then eliquate and take with it from the hotter zone below less fusible parts of the charge, thus forming prematurely the intended slag. If, in charging, the ore is followed by a flux of similar composition, very liquid slag cannot form in the upper parts of the furnace, but slag requiring a higher temperature for its formation will be formed lower down, and will combine with the liquid slag at the tuyeres and form the normal slag. Thus, for instance, a silicious ore should be followed by a silicious iron flux, and the dissimilar flux—limestone—come last. In this way a higher temperature will prevail in the smelting zone than would if ore and fluxes were mixed and fed together into the furnace. Henrich explains the good effect large charges often have in furnaces as compared to small ones, by the separate charging of ore and fluxes with the former, while with the latter, ore and fluxes are usually dumped in one heap in front of the feed-door and charged together into the furnace. They have also another advantage in that they reduce the amount of fluedust formed. The normal weight of a charge was formerly 1,000 lb.; to-day three such charges are fed at once with the most beneficent effect on the yield of lead. As a rule there is no advantage in feeding heavier charges, unless of course a furnace is exceptionally large.

A furnace is in a good condition on top when this is cool, when the charges sink regularly and evenly, and the fumes ascend uniformly, or a little more densely at the sides than at the center.

The labor required on the feed floor varies with the size and number of the furnaces and the conveniences for handling materials. A furnace from 33 to 42 in. between the tuyeres and from 84 to 120 in. in length requires per shift one feeder and from two to three wheelers; with 10 furnaces under the same conditions are required 10 feeders, but the number of wheelers will not increase in the same proportion. Both generally work 12-hour shifts, although sometimes the feeders work only eight.

The tools required by the feeder are a square-pointed, long-handled shovel, and a scoop or fork for the fuel, a 6-lb. napping hammer for breaking up the coke, and a stiff broom, several heavy 1½-in. steel bars of different lengths, the longest ones reaching from the top of the jackets up to 4 ft. above the feed floor, and several 10-lb. double-faced sledges. The bars and

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sledges are used in cutting out wall accretions (§ 77), and one set of tools is sufficient for a number of furnaces. The wheelers require several iron wheelbarrows for ore and fluxes and a wooden barrow or buggy for fuel, several round-pointed shovels, a scoop or fork, several coarse brooms, and generally a pick. A definite number for most of the tools cannot be given, as it varies too much with the general arrangement of the feed floor.

§ 75. Regular Work on the Furnace Floor.—This consists in regulating the water supply, taking care of the tuyeres, tapping speise, matte, and slag all together or speise and matte together and slag separately into slag-pots and wheeling them to the dump, tapping the lead either first into the lead-pot or ladling it directly into the molds, and piling the bars of base bullion upon the ground or on a truck.

The water of the jackets is kept at about 70° C., the usual test being that the hand can be quickly passed through the outflowing water without being scalded. Any irregularity in the temperature of the jackets indicates that the coating on the inside is thicker in some than in others, and thus that the smelting proceeds unevenly; the slower descent of the charges on the cold side will corroborate this on the feed floor.

The pressure of the blast is constantly watched and all the changes in the gauge noted. It is regulated by means of a damper. If the furnace has its own blower, its revolutions are also counted.

The appearance of the tuyeres forms a good indication of the inner condition of the furnace. It is not necessary that the entire mouth of the tuyere should be bright; it is often covered by a thin scale of slag, showing a star-like brightness in different places. If it becomes quite dark, a rod is inserted and the slag pierced to see if it has grown too thick. In this case repeated poking only aggravates matters, as a "nose" of chilled slag forms, reaching into the furnace. (For correction of this evil see § 77.)

With furnaces having an internal crucible, the slag is tapped at intervals; if the crucible is partly internal and partly external, the slag will flow continuously. The tapping of the slag is done with a square-pointed steel bar. The slag runs over the slag-spout into the slag-pot; when full, the tap-hole is stopped with the stopping-rod. As regards liquidity, matte comes first, then speise, and last the slag. When flowing from the spout they can

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be readily distinguished. The clay stopper in the tap-hole ought to be sufficiently soft to be perforated by pricking it with the bar, but oftener it requires a few taps or strokes with a sledge. With a hard tap it is sometimes advisable to insert a small piece of charcoal into the clay stopper so that it protrudes slightly. On shutting off the slag, the charcoal enters the opening made and is followed by the clay. On tapping again very little sledging will be necessary. The use of this charcoal is kept up until the correction in the charge or the feeding has softened the tap. It is a rule always to keep some slag in the furnace. At certain intervals, however, the lead and slag are allowed to rise until the little blue flame, seen through the tuyeres, indicates that the slag has nearly reached that level, when it is all tapped out to see how many pots of slag the furnace holds. If there is less slag than usual, it shows that the region of the tuyeres is not as free as it ought to be. If a furnace is making so little speise and matte that they need not be separated, they are usually tapped with the slag. It is then generally desirable to accumulate the speise and matte in as few slag-pots as possible, as, when they run out with the slag, they are splashed, especially the matte, against the cool sides of the pot. The separation of matte and slag being complete only near the center of the pot, where they remain liquid for some time, an outer ring of rich slag, assaying often 15 oz. and more to the ton, will be obtained when the bulk of the slag is poor. This occurs especially with acid slags; with slags that are very fluid the difference in outer and inner slag is not so great. If the furnace has a clay breast, the slag is tapped when it has accumulated in the furnace a little below the ordinary tap-hole. All the speise, if any is made, and matte, and perhaps a little lead, will run out and be collected in one pot, and all the slag of this pot will be saved. After a number of tappings from the ordinary tap-hole, regulated by the speise and matte the furnace is making, the tapping lower down is repeated. If the furnace has a water-cooled cinder-notch (tapping jacket), the level to which the slag is allowed to rise is regulated by the lead alone, as a jacket with two tap-holes is not much used.

If a furnace is making much matte, collecting-pots, such as those described under § 70, will effect a more or less satisfactory separation outside of the furnace, or matte (and speise) may be removed from a separate tap-hole, as is always the case with fur-

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naces having a crucible that is partly external and partly internal and occasionally if the crucible is internal.

It is of importance for the good running of a furnace that the level of the lead should be kept high. If too low, the speise and matte will be too far removed from the zone of fusion, and thus liable to chill and form a crust. The less lead there is in the charge the higher must the level of the lead in the crucible be kept, that it may not cool, as it is not frequently replaced by fresh lead that has just passed the hottest part of the furnace. With a charge containing from 7 to 9% of lead, the lead in the well is kept often as high as 8 and 10 in. above the slag-tap. That it may be prevented from breaking through, a small square cast-iron frame with tap-hole is placed on the crucible and lined with brick. The bullion used formerly to be taken out altogether by dipping, but if this is not done with judgment too much is apt to be removed, which causes trouble. The amount is to be regulated by the number of charges that have been fed into the furnace. In ladling the hot lead from the basin of the well into molds, most of the dross held in solution rises to the surface when the bar cools, and is removed by skimming, but some always remains. This dross often causes dispute between buyer and seller as to the silver contents of a given shipment of bullion. The dross assays much lower in silver than the clean lead, and as it is not uniformly distributed in the bar, the sampling becomes irregular (§§ 82 and 83). Then it is the pride of the furnace-keeper to produce bars with a clean surface. This has led to pouring the mold one-quarter full of lead, then adding dross, and covering it up with a layer of clean lead. The trouble can be done away with by allowing the lead to accumulate in the well, and tapping it at intervals into a cast-iron pot. The lead, not being dipped out, cannot sink lower than the level of the tapping-hole, which is on a line with the wall of the crucible, and can be kept as high as desired. In the cast-iron pot the dross is skimmed off, and the cooled lead is then ladled into a series of eight or ten molds placed in a row alongside the furnace on two rails, supported by two wooden horses. Nevertheless, many metallurgists prefer to dip the bullion from the lead-well. The reason is that, if the lead is tapped at long intervals from the well, a large number of bars, from 10 to 14, will be taken out of the furnace at once, and the charge will sink suddenly to occupy the

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space set free by the lead, with the result that particles of unmelted charge pass too quickly through the melting zone to be fused, and often form the nucleus for a crust. They therefore dip the lead from the well as fast as it forms; *i.e.*, the dipping is regulated by the number of charges fed into the furnace and the percentage of lead it contains. Dipping and tapping the lead are both used; the dipping may perhaps be preferable where charges run as low in lead as 8%. At some smelting works, no matter in whatever way the base bullion has been taken from the blast furnaces, it is purified before it is shipped to the refiner. This is done by melting it down in a kettle holding about 20 tons of lead, skimming off all the dross that rises to the surface, and then siphoning the liquated bullion, as is done with the refined lead from the merchant kettle (§ 116), into molds and taking a liquid sample from the siphon for every tenth or fifteenth bar molded. In this way the smelter ships clean bullion, saves the trouble of sampling by punching or sawing, and knows just how much lead and silver he is shipping; the dross goes to the ore or matte-charge, where it belongs, to be converted into lead and matte, and not to the refiner who can afford to make a lower refining charge for liquated than for drossy bullion.

When the bars have solidified they are marked with the running lot-number, removed from the molds with a pick, and piled up to be sampled (§ 83), if this has not already been done, weighed, and shipped.

A furnace is in good working order below when the temperature of the jackets is uniformly high, the pressure of the blast does not fluctuate, and the tuyeres remain bright, having only short "noses." The furnace should within a given time produce always the same number of pots of slag, the tap-hole be neither too hard nor too soft, and the lead in the well be of a bright-red color, play with the blast, and sink slightly every time a pot of slag is tapped.

On the furnace floor are required for every furnace (from 33 to 42 in. by from 84 to 120 in.) one furnace-keeper, one tapper who looks after the lead, and from one and one-half to three pot-pullers, according to the size of the dump and the manner of disposing of the slag.

The tools required by the furnace-man are two tapping-bars (6 or 8 ft. long, of $\frac{3}{4}$-in. steel), two stopping-rods (10 ft. long, of

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½-in. iron, with a disk 2½ in. in diameter for the clay plug), two iron rods (12 ft. long, of ¾-in. iron) for special cases, three or four heavy (1 to 1¼ in.) steel bars about 6 ft. long, and two 10-lb. double-faced sledges; a 2-in. steel bar 8 ft. long is handy, if a crust is to be pried up. The tapper's tools are few: a ½-in. steel bar, 3 ft. long, a cast-iron ladle (6 in. in diameter, 3 in. deep) riveted to a 4-ft. handle, a pick, a skimmer to clean the well, a set of steel dies, a 4-lb. hammer to number the bars with ¾-in. figures, and a chisel to trim them. The pot-pullers have to mop the pots with a clay wash; they require no special tools, excepting occasionally a steel bar and sledge to loosen a cake of matte or slag from the slag-pot, should it adhere firmly.

§ 76. Work on the Dump.—If the ordinary slag-pot is used, it is wheeled out on the dump and allowed to cool. When the contents have solidified the pot is tilted, and the cone of slag, with speise and matte adhering to the bottom, rolled out. This is broken up with a sledge, matte and speise are sorted out and piled up, and the clean slag is thrown over the dump. Sometimes the slag is loaded into flat-cars in pieces as the result of breaking up the cones, or else the clean slag is first put through a crusher, thus preparing it for the direct use of the railroad as ballast.

When the overflow-pot is used which discharges slag free from shots of matte into an ordinary slag-pot, it is allowed to cool on the dump. The contents of the other pot are emptied while still liquid over the edge of the dump. The scale of slag forming on its sides is reserved, if required for the blast furnace. With a Devereux catch-pot the pot is wheeled either to the edge of the dump and the liquid slag is discharged there, or to the large side-dumping pot, if this be in use. The tap-hole of the catch-pot is opened from the inside by thrusting a steel bar through the crust of the slag. The large tilting-pot is generally placed in a trench left open in the dump, and receives there from both sides the liquid slag. When filled, it is hauled to the edge of the dump by a horse or mule and emptied. Occasionally this receiving-pot is placed between two furnaces to receive their clean slag, and hauled, when filled, by a wire rope to the edge of the dump and there tilted.

All the slag remaining in the overflow-pot or in the catch-pot with tap-hole goes back to the blast furnace. The manipulation

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of the Livingstone settling-pot has already been described, as well as the granulating of slag (§ 67).

The dragging-out of slag-pots to the edge of the dump is greatly facilitated by keeping its surface even and smooth. This is done by forming 3-ft. squares with rails, old castings, blocks of slag or sand on top of the rough surface and filling them with liquid slag. Keller* prefers to make oblongs 2 by 4 ft. and from 8 to 12 in. deep, to fill them partially with large pieces of cold slag and cement these with liquid slag poured at the same time from several pots. Covering part of the dump with cast-iron plates, as done by Livingstone at the Omaha & Grant Works, Denver, Colo., gives the best surface. Although the first cost was considerable, it has more than paid for itself.

In large works the dump grows very quickly in width, if the slag is not taken away. The tramming of the waste slag to the edge in large receiving-pots is usually done with a mule or horse, occasionally with a stationary hauling engine. Austin† has proposed the following method for the disposal of slag. The car carrying the waste slag is raised and allowed to run by gravity to a distant point, where it turns on a curve and, still descending, comes back on the edge of the dump, where it is emptied; when empty it is run further down the inclined plane to the place where it is again to be filled.

The number of men required on the dump varies with the size of the furnace and the manner of disposing of the slag. Three men in a 10-hour shift can clean up the slag and matte made by a furnace 42 by 120 in. at the tuyeres. This represents good work. The tools required are a steel bar, a pick, a sledge, a round-pointed, long-handled shovel, and an iron wheelbarrow.

§ 77. IRREGULARITIES IN THE BLAST FURNACE.‡—The disturbances that occur during the run of a blast furnace, having many different sources, are numerous. Some are caused by defective machinery and apparatus, others by refractory ores, faulty charges, or wrong manipulation. Some accidents due principally to the last-mentioned cause may be corrected in the following way.

When the charges do not descend as evenly as they should, one

* "Transactions American Institute of Mining Engineers," xxii., p. 576.

† *Ibid.*, xxvi., p. 401.

‡ Lang, *Engineering and Mining Journal*, Dec. 20, 1893; Jan. 20, 1894.

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side sinking faster than the other, the jackets on the lower side being much hotter and the tuyeres brighter than those on the upper side, the first change is made in the manner of feeding. The fuel is placed more on the hanging side, and ore and fluxes on the quickly-descending one; then a full stream of water is turned into the hot jackets and that of the cool jackets reduced till there is just an overflow. By the combination of these remedies the smelting of the furnace on one side more than the other will be corrected and the charges will right themselves again after a few hours. Shaking up the charge with a long, heavy (1½-in.) steel bar, introduced through the feed-door into the hanging side, will often hasten matters. At some works that have to treat ores rich in zinc, it is the practice at the beginning of each day-shift to drive in a steel bar at the four corners of the furnace as far as the top of the jackets, thus loosening any wall accretions that are forming. By doing this, the number of times a furnace has to be barred down to remove accretions is greatly reduced and the whole running improved.

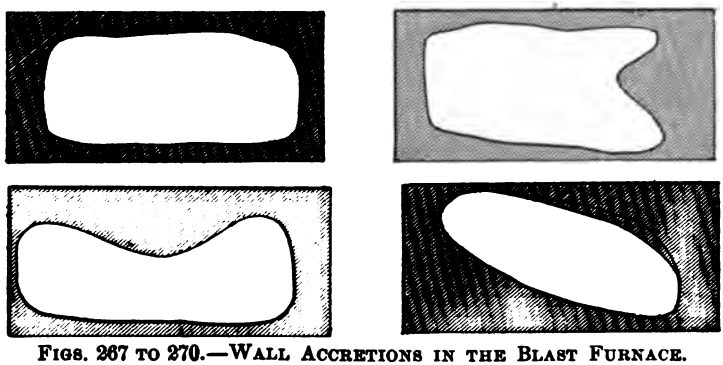
When the charges descend irregularly, it often happens that the fire creeps up and it becomes hot on the surface (overfire, fire-tops, hot top). The furnace may then be fed down, which consists in adding only just enough fresh charge to keep the flame or heavy smoke from passing through. When the surface has thus been lowered for 2 or 3 ft., the furnace is filled up quickly again and the top thus cooled. Simple sprinkling of water on the top of the charge has only a temporary effect. This feeding down helps matters if the over-fire does not come from a crust in the crucible.

The cause of the irregular descent of the charges lies generally in the formation of wall accretions (§ 90), which begin on top of the jackets and grow thicker toward the feed-door. They assume different forms. The following figures, 267 to 270, by Iles* show some extremes. Fig. 267 represents a more or less regular shape, and the smelting power of the furnace is not necessarily reduced. With irregular hangings, like those in Figs. 268, 269, 270, the descent of the charges will be greatly obstructed and the amount of fluedust much increased. The charge will be tight at the narrow parts of the furnace, and the blast entering the tuyeres will be concentrated in a few places

* *Engineering and Mining Journal*, Feb. 6, 1886.

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and cause "blowholes." As soon as these wall accretions are discovered they have to be cut or barred down. In order to reach the lowest part, the furnace is fed down and the blast lessened at the same time, till the charge has reached about the top of the jackets. While it is being lowered no lead is removed from the well, in order that the crucible may be entirely full while the barring out is going on. The blast is stopped, the blast-pipes are removed, the tuyeres closed, all the slag in the furnace is tapped, and the flow of water into the jackets nearly shut off. When everything is ready a charge of coke is given, with some slag and flux, forming a bed for the accretions to fall on as they are chipped from the walls. The cutting-out is best begun just above the jackets. A square-pointed steel bar about 1¼ in. in diameter and long enough (about 18 ft. to reach from



FIGS. 267 TO 270.—WALL ACCRETIONS IN THE BLAST FURNACE.

the top of the jacket well into the opposite side on the feed floor, is driven with a sledge into the crust above the jacket. If it does not yield, a rope tied around the head of the bar is thrown to the opposite side, where several men pull it and thus break off the crust. This is repeated until it has been removed in a number of layers, two sets of men working on opposite sides. While the crust is being barred off, fuel, slag, and fluxes are added that it may be smelted out, when the furnace is started up again. The reason that the barring down is begun from below is that otherwise the broken crusts and small slag-charges that have been added might so fill up the furnace that it would be impossible to reach the crust at the top of the jackets, and new accretions would form again quickly. Sometimes, however, if the crust is

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thick and hard, the barring-down is begun from the top, continuing until the accumulated crust and slag-charge meet the clean side-wall, when it is smelted out, keeping the charges low. The furnace is now let down again, and the second half of the barring-down is begun at the top of the jackets and continued upward.

When the sides of the furnace have been cleared, the tuyeres are cleaned out, the tuyere-pipes inserted, a weak blast, to be increased gradually, is turned on, and smelting resumed. Care must be taken about the water supply as the jackets grow hotter, and about the tapping of the slag, as the wall accretions often melt very fast, and there is danger of the slag entering the tuyeres. After starting the furnace it is sometimes found that a small crust has formed over the lead while the blast was shut off. This is perforated with a long iron rod or with a steel bar, if necessary, and will soon disappear, if the furnace was in good working order below before the cutting-out began.

A second method of barring down a furnace, said to work well, is to let down the charge to below the upper rim of the jackets, and give a bed of coke as previously described. In the meantime, the bricks between the jackets and the collar, on which the shaft rests, are removed for a distance of 1 or 1½ ft. The accretions are cut down, beginning from above, and raked out through this opening. When the shaft is clean, the opening is bricked up, light charges are given, the furnace is filled up with ore-charge, and the blast let on. The accretions go to the roasting furnaces.

A third method of barring down is dangerous, although it may be necessary at times and is even exclusively used by some metallurgists. The charge is let down to the top of the jackets, and cutting out begun from the top and continued until the crust collecting in the furnace has risen so high that the work cannot proceed any further. The two breast-jackets are now taken down and the contents of the furnace raked out (§ 78), in order that the cutting may continue until the walls are clean. This may take as much as 18 hours. Two parties of three men each, working on opposite sides of the furnace, do the cutting, one man holding the bar and two sledging. As the work is hard and has to be done as quickly as possible, the regular hands have the constant assistance of furnace-men and helpers from other fur-

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naces, who work half an hour at a time. While the accretions are being removed, a crust 6 in. thick or less will form on top of the lead. This is broken up, the breast-jackets are put back, and the lower front is closed. Charcoal is fed from above to the top of the jackets and the furnace blown in anew, adding 25 bars of lead to the first slag-charges to heat up the lead in the crucible.

Guyard* suggests that by using caustic lime in the charge instead of limestone these accretions might be made less troublesome, as lime has a decomposing action on the sulphides, of which they consist in great part. The use of burned lime as a basic flux has already been commented upon (§ 67). As to its desulphurizing action it should be remembered that while accretions next to the furnace walls consist mostly of sulphides, they are usually covered by a thin crust as hard as flint; this is followed by a softer substance that is often a powder, and this again covered by a crust so tough that it is sometimes extremely difficult for a steel bar driven hard with a sledge to produce any effect on it. It does not seem probable, therefore, that caustic lime would have any important effect on these composite crusts, which must differ from sulphides as much in their chemical properties as they do in their physical.

It often happens, even when a furnace is otherwise doing well, that the tap-hole becomes hard and the tuyeres dark. The fuel is then distributed more over the cold places, additional fuel being given for a short time if necessary. One or two bars of lead are sometimes placed on top of the charge over the tap or the tuyere, but this cannot be commended, although it is often effectual. The change in feeding the fuel will generally soften the tap. To soften a crust in front of a tuyere, the bag or the gate should be closed, or nearly closed, as the blast playing on chilled slag can only have a bad effect. If it is turned off, the heat generated at the neighboring tuyeres will melt off the crust; then a little blast is allowed to pass through the tuyere, and gradually increased until the full blast can be turned on again.

Passing a tube through this† and thereby conducting the blast safely through the crude ore may help matters. To remove the fine ore Lang‡ shuts off the blast from the tuyere and inserts a

* Emmons, "Geology and Mining Industry of Leadville," pp. 728, 747.
† Lang, *Engineering and Mining Journal*, Feb. 8, 1890.
‡ *Loc. cit.*

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tube in the ore through the tuyere-pipe, when the inside pressure will blow out most of the fine ore, which can be assisted by working the tube. This proceeding may also be necessary if some fine ore trickles through the coarser parts of the charge, appearing in a crude state at the mouth of the tuyere, and, as is sometimes the case, runs into the tuyere-pipe. If fine ore appears at the same time at several tuyeres and too many would have to be tied up, the easiest remedy is to feed down the furnace and thus loosen the charge. If this does no good, a coarse ore-charge will have to be substituted for a short time, or if coarse ore is not to be had, a slag-charge.

The forming of hearth accretions (§ 91) is indicated on the feed floor by the top becoming hot and the charges not sinking regularly, but in jerks, a foot at a time, and blazing up with every settling. On the furnace floor the lead in the well becomes dark, and does not play with the blast. By inserting a rod through the tap-hole, the position and often the thickness of the crust can be ascertained. Sometimes it only forms a ridge across the furnace, and communication is open in front and at the back; more commonly the crust begins at the back and grows toward the front, gradually closing all communication. The lead produced in the furnace cannot get into the crucible, and runs out with the slag. A case like this generally needs a change in the composition of the charge; the slag analysis will show the defect and give the remedy. As it takes from six to eight hours before a new slag makes its appearance, holes are driven with a steel bar through the crust that the lead may find its way into the crucible. If the crust has not yet hardened, it is repeatedly perforated by introducing an iron rod and working this up and down and to both sides. The crust may be only a temporary affair, and can then be worked into the slag, which removes it. Cases do occur where the crust will not yield quickly enough to being fused out and has to be removed by force. This has been ironically spoken of as "muscular smelting," but it is sometimes unavoidable. Before beginning, all the slag in the furnace is tapped, the basin of the lead-well covered with glowing charcoal, the blast stopped, the tuyere-pipes drawn out, the tuyeres closed, and the flow of water into the jackets reduced. The breast of the furnace is now removed wholly or in part, any chilled slag in front is chipped off, and some of the loose material raked out

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into a wheelbarrow in front of the furnace and taken away. A heavy steel bar is passed through two opposite tuyeres nearest the breast to hold up the charge. Balls of loam or clay are tamped behind the lower part of the front jacket to prevent the charge from rolling down. The balls, placed on the end of a board, are slid below the front of the jacket and rammed upward against the charge with a rod bent to a hook. When this is done, any lead that has accumulated in front is ladled out, and the crust thus laid bare. A hole is driven through it with a heavy steel bar. It is necessary to have a number of these ready, as the points soon become dull or bent. The hole is enlarged by driving the bar again close to it and breaking the crust toward it. When large enough to receive a 2-in. steel bar, this is warmed, inserted into the hole, the crust pried up, and the broken pieces raked out. If the crust will not yield, the hole is enlarged and the furnace started up again. The lead previously bailed out is returned, and, if necessary, fresh hot lead added, the clay balls are removed, the hollow space in front is filled with charcoal, the steel bars are withdrawn from the tuyeres, the breast is put in, the tuyeres are opened, the pipes inserted, and the blast is let on, but very gently at first. When the first two or three pots of slag have been tapped the rod is repeatedly inserted to keep the hole open until the new slag comes down. The lead will soon show the effect of having communication between it and the slag partly restored; it begins to play with the blast and becomes hotter, thus assisting the work of the new charge.

A crust is sometimes caused by a leaking jacket. This is first indicated by the appearance of moisture at the tuyere or the bottom of the jacket. The leak, if small, can be temporarily stopped by mixing cornmeal with hot water, pressing it with the hand into small balls, and throwing these into the water-feeder of the jacket. Soon, however, the jacket will have to be removed. For this purpose the crust on the inside is first allowed to grow thick by cooling, which is done by turning in a full stream of water, and opening the discharge at the bottom. Two courses of brick are chiseled out above the jacket. When cool, the furnace is stopped, the cooling-water on the side of the injured jacket shut off, the water-trough removed, and the injured jacket unhinged, taken out, and a new one put in its place. The whole

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procedure need not take more than 20 minutes. Should the crust on the inside of the jacket prove too thin and break out, the opening is closed by the introduction of clay balls. The space where the new jacket is to be inserted must be absolutely clean, as any little pieces of brick or other hard matter will obstruct the placing of the new jacket and cause much delay. Of course the foregoing has no reference to a furnace with wrought-iron jackets extending its entire length.

The clogging up of the lead-well is yet to be considered. In smelting sulphide ores rich in lead, sulphide of lead held in solution in the crucible often separates out when the lead ascends the channel toward the basin. A bent iron rod may be inserted to clean it out. This presence of sulphides is generally caused by an incomplete decomposition of galena in the furnace on account of lack of heat in the smelting zone. If the charge is rich in copper, this causes coppery lead gradually to close up the channel.

§ 78. Blowing Out.—If ore, flux or fuel gives out, or if the furnace needs to be repaired, or if an accident happens that cannot be remedied in a short time, say in 18 or even 24 hours, the furnace has to be blown out. This is done by stopping the ore-charges and substituting slag-charges, until most of the ore has been smelted out. The charge is allowed to sink and the blast is gradually lowered. Soon volumes of dark smoke mixed with white lead-fumes will appear. When the charge has receded somewhat, and before a flame makes its appearance, the damper in the flue leading to the dust-chamber is closed, and the fumes are conducted into the air by opening the damper on the top of the furnace or by lowering the sheet-iron stack, or by whatever contrivance may be in use for the purpose. If this were not done, an explosion might occur in the dust-chamber. To check the flame and to reduce the temperature, water is often sprinkled over the charge, although its effect on the lining of the furnace cannot but be deleterious. When the charge has sunk as far as the top of the jackets, the blast is stopped and the tuyere-pipes are removed. All the liquid slag is tapped, the tapping-jacket removed, and the breast of the furnace is knocked in.

Sometimes the furnace is blown down, allowing the charge to sink till only heavy fumes, but no flames, appear and the entire

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contents are then drawn. In this case there is no need of closing the damper to the dust-chamber; in fact, many furnaces have no damper at all.

The bulk of the slag remaining in the furnace is withdrawn with a hoe into iron wheelbarrows, which are emptied on the dump and then chilled with water. As it is important that there should be little delay in drawing the charge, four or five wheelbarrows are placed one behind the other near the front of the furnace. As soon as the first is filled with red-hot charge it is wheeled away and replaced by the second, the emptied wheelbarrow being put at the end of the line. When all the charge that can be easily reached with the hoe has been drawn out, the front jackets are taken down and the rest removed. Meanwhile a thin crust will have formed on top of the lead in the crucible. This is easily broken, and the lead is then ladled out into the molds that have been moved from the lead-well to the front of the furnace. Blowing down is more expensive than is generally believed. The items to be considered are: fuel remaining in the furnace, loss of half a shift in blowing out, dipping the bullion, barring out, sampling products, treatment of barrings, work of mason and extra firebrick and fireclay, warming the crucible, fuel for blowing in, and loss of half a shift in blowing in. Beside, the general expense of the works goes on in the same way, whether all the furnaces are working or only part of them. If a furnace has been blown down and is liable to be blown in again in a short time, it is advisable to keep the crucible warm, and a pole, 20 or more feet long, is inserted from the breast and kept burning slowly.

§ 79. Furnace Books.—A daily record of the work done is kept for each furnace. One of the many suitable skeletons for this purpose is shown in Table A, following. The time when any change in the charge is made, or when anything out of the regular way occurs is noted under the first column of "Remarks." The second column of "Remarks" refers to the shipment of the bullion.

§ 80. Furnace Assay Book.—A separate book is kept to record the assays and analyses made of slag and matte (see Table B, on next page). Under the head of "Remarks" are brought the names of any of the other furnace products that may be assayed now and then.

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TABLE A.

189– November.	Furnace, No.	Ore-Bed, No.	Charges, No.	Composition of Charge, Pounds.						Blower.		Remarks	Total Pounds of Charge.						Output.						Remarks.
				Ore.	Iron Ore.	Limestone.	Slag.	Coke.	Charcoal.	Rev., No. Per Min.	Pressure, Ozs.		Ore.	Iron Ore.	Limestone.	Slag.	Coke.	Charcoal.	Pots of Slag, No.	Bars, No.	Lot, No.	Lb.	Ag, Oz. Per Ton.	Au, Oz. Per Ton.	

TABLE B.

Date.	Furnace, No.	Slag.										Matte.					Remarks.
189– November.		Per Cent.									Ounces Per Ton.	Per Cent.				Ounces Per Ton.	
		SiO_2.	FeO.	MnO.	CaO.	BaO.	MgO.	Al_2O_3.		Pb.	Ag.	Pb.	Cu.			Ag.	

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FURNACE PRODUCTS.

§ 81. FURNACE PRODUCTS.—The products of a blast furnace are base bullion, speise, matte, slag, wall accretions, hearth accretions, furnace cleanings, and fluedust.

§ 82. BASE BULLION is the commercial name for argentiferous lead, as distinguished from silver or gold bullion. It is cast in bars weighing from 80 to 110 lb. and contains in addition to the silver and gold, small quantities of other metals, as seen by the following analyses:

	Clausthal.	Sophienhütte.	Freiberg.	Mechernich.	Pribram.	Leadville.	Leadville.
Pb	98.80944	99.641096	95.088	99.5913	97.3597	99.0796310	98.492379
Ag	0.1412	0.000250	0.470	0.0215	0.4230	0.6119445	0.793417
Bi	0.0048	0.352053	0.019	.....	0.0070	Trace.	0.011791
Cu	0.1862	0.000279	0.235	0.1382	0.1100	0.0479100	0.071450
Cd	Trace.	.....	.....	.....	.....	Trace.	Trace.
As	0.0664	.....	1.826	.....	0.2900	(?)	(?)
Sb	0.7906	0.002872	0.958	0.2180	1.5240	0.2138940	0.347881
Sn	.....	.....	1.354	.....	0.2500	Trace.	0.000897
Au	.....	.....	.....	.....	.....	0.000888	0.000891
Fe	0.0664	0.002877	0.007	0.0300	0.0096	0.0063000	0.012600
Zn	0.0028	0.000673	0.009	0.0060	0.0012	0.0016052	0.000232
Ni	0.0033	.....	.....	Trace.	0.0015	.....	.....
Co	0.00016	.....	.....	.....	.....	.....	.....
S	.....	.....	0.051	.....	0.0300	.....	0.048934
References	(a)	(b)	(c)	(d)	(e)	(f)	(f)

(a) *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xviii., p. 208. (b) *Ibid.*, xix., p. 169. (c) *Wagner's Jahresberichte*, 1887, p. 401. (d) *Berg- und Hüttenmännische Zeitung*, 1886, p. 434. (e) *Oesterreichisches Jahrbuch*, xxxix., p. 52. (f) Emmons, "Geology and Mining Industry of Leadville," p. 694.

The impurities of a bar of lead, with the exception of silver and gold, always collect nearer the top than the bottom. This is illustrated by the analyses of Streng, made from non-argentiferous lead, and by Schertel, made from base bullion.

	Streng. (a)				Schertel. (b)	
	Top.	Bottom.	Top.	Bottom.	Top.	Bottom.
Ag	.....	.....	.....	.....	0.423	0.403
Bi	.....	.....	.....	.....	0.132	0.042
Cu	3.621	1.242	0.508	0.140	1.324	0.034
As	.....	.....	.....	.....	2.164	1.980
Sb	0.274	0.158	0.090	0.057	0.700	0.749
Sn	.....	.....	.....	.....	0.941	(?)
Fe	0.008	0.008	0.012	0.008	0.103	0.009
Zn	0.008	Trace.	0.002	Trace	0.016	0.003
Ni	0.148	0.082	0.012	Trace	0.029	.....
S	.....	.....	.....	.....	0.500	.....
Sp. gr	.....	.....	.....	.....	10.321	10.324

(a) *Berg- und Hüttenmännische Zeitung*, 1859, p. 14. (b) *Wagner's Jahresberichte*, 1887, p. 401.

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Schertel's analyses would seem to show that silver has a similar tendency. He, however, took his sample from lead that had been kept for 24 hours above its melting-point in an iron cylinder, 3 ft. $3\frac{3}{8}$ in. high, and thus created special conditions. In bars of base bullion which cool quickly just the reverse is the case. The following tests made by the writer in 1881 may substantiate this. (Figs. 271 and 272.)

Bars of base bullion were sawed into three pieces and samples taken as shown by the numbers and assayed separately.

Fig. 271.	(1.) 149.7	(2.) 150.5	(3.) 146.0	(4.) 148.7	(5.) 145.0
	(6.) 187.0	(7.) 152.0	(8.) 149.0	(9.) 149.0	(10.) 151.0
	(11.) 148.0	(12.) 150.0	(13.) 150.5	(14.) 150.0	(15.) 152.0
Fig. 272.	(1.) 127.0	(2.) 134.5	(3.) 128.5	(4.) 129.5	(5.) 125.0
	(6.) 133.5	(7.) 124.0	(8.) 132.0	(9.) 126.5	(10.) 134.5
	(11.) 129.0	(12.) 132.0	(13.) 133.0	(14.) 132.0	(15.) 134.0

The results (ounces per ton) clearly prove that the whole lower part of the bar is richer than the upper and that the center of the upper surface is the poorest part. Similar results were ob-

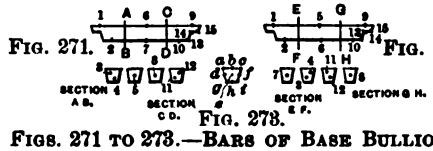


FIG. 271. FIG. 272.

FIG. 273.

FIGS. 271 TO 273.—BARS OF BASE BULLION.

tained by Piquet* and Raht.† Raht shows that that part of a bar which solidifies last is the poorest in silver. The bar represented in section in Fig. 274 was sprayed with water as soon as molded, that in Fig. 275, cast from the same bullion and at the same time, was allowed to cool in the usual way. In Fig. 274 the shell, cooled by mold and spray water, is the richest; in Fig. 275 the parts in contact with the heat-withdrawing iron are the richest.

An attempt has been made to explain‡ this concentration of the silver in the lower part of the bar by the separation of the argentiferous lead, while cooling, into crystals low in silver and liquid lead high in silver. It is said that when a bar has been molded, the surface cools first, and crystals begin to form there; that the sides then slowly solidify, a mass of liquid lead and crystals remaining in the center. As the cooling is supposed to proceed

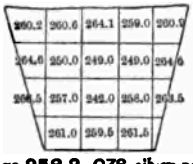
* Roswag, "Désargentation du plomb," Paris, 1884, p. 127.

† "The Mineral Industry," iii., p. 414.

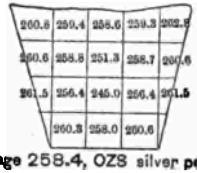
‡ Roswag, *Op. cit.*, p. 126.

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from the top downward the crystals will continue to form on the cooler upper side and gradually force the richer liquid lead toward the bottom. This theory presupposes that the mold is a poorer conductor of heat than the air, which cannot be the case. The reason that the surface of a bar is so much poorer than the



Average 258.2, OZS silver per ton.



Average 258.4, OZS silver per ton

FIGS. 274 AND 275.—SECTIONS THROUGH BARS OF BASE BULLION.

other parts is not only because of the uneven cooling, but also on account of the impurities that rise to the surface. They run much lower in silver than the pure lead, as is shown by the following assays.*

BASE BULLION ASSAYS.

Weight of Lead. Oz.	Weight of Dross. Oz.	Assay of Lead. Oz. Per Ton.	Assay of Dross. Oz. Per Ton.
261¼	10¼	87.82	74.04
283¼	9½	92.05	74.47
273¾	7¼	86.59	68.55
65	15½	84.21	66.47
311¼	11¼	94.84	77.30
30	10	87.79	74.40
283¼	7¾	98.30	75.56
233¼	10	95.96	76.44
241¼	11½	105.15	87.59
32	11½	89.62	77.10
54	24	82.91	67.56
511¼	28	84.13	70.00
532½	14	106.94	78.92
45	17	87.24	68.80
47	12	98.69	79.25

If bullion is so rich in dross that the lead cannot eliquate freely, the natural distribution of the silver will be much disturbed. This will account for the fact that assays from the top of a bar sometimes run higher than those from the bottom. For instance, coppery bullion from the Ramshorn silver mine, Idaho, containing so much dross that if dropped on the floor it would break, gave to Rhodes† the following results in ounces per ton (Fig. 273), a. 432, b. 432, c. 439, d. 450, e. 434, f. 441, g. 385, h. 397, i. 386.

* Private communication by W. Tatham.
† Private communication, July, 1891.

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According to Kempf, Nenninger & Co.,* gold also seems to be concentrated with the silver near the bottom of the bar.

Sample From	Ag.	Au.	Ag.	Au.
Top.....	143.3	4.59	129.7	3.68
Bottom.....	148.4	4.76	134.3	3.89

Similar results have been published by Torrey and Eaton.†

§ 83. SAMPLING.—The irregular manner of sampling and the unequal distribution of the silver have caused much trouble in estimating the value of base bullion. Samples used to be taken, for instance, with a gouge from top and bottom near the opposite ends of a bar. This method has two sources of error, the form of the sample and the fact that they are taken only at the ends. The sample obtained is a conical chip having its base at the surface and its apex $\frac{1}{2}$ in. or less below. The incorrect form has been changed by using a punch which gives a cylindrical chip about $\frac{1}{8}$ in. in diameter. Such a punch is represented in Figs. 276 and 277. At a distance of $1\frac{1}{16}$ in. from the point it has a shoulder so that, when driven vertically into a bar, it will penetrate 2 in., or halfway, with the shoulder making an indentation of $\frac{1}{16}$ in. It will take an approximately correct sample of that part of the bar. If driven halfway through the bar, say 2 in., it will represent a correct sample of that part of the bar. Sampling, however, from the ends of the bar gives too high a result, as can be seen by turning to Figs. 271 and 272. The poor parts of the bar, represented by No. 6 in Fig. 271 and No. 5 in Fig. 272, are excluded. The common method to-day is to take punch samples diagonally across a row of, say, five bars, as shown in Fig. 278, by driving the punch every time halfway through, turning over the bars and repeating this on the other side in the opposite diagonal. In order to show where the holes are to be punched it is advisable to use a templet of sheet iron fitting over the edges and having little spring punches which when tapped will mark the right spots on the bar. It will be noted that none of the spots fall either on the edge or in the median line. Two men usually work together, using 4-lb.

* *Engineering and Mining Journal*, July 1, 1882.
† *Ibid.*, Dec. 25, 1886.

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sledges. One man will hold the punch vertically, while the other gives it the first blow, then both strike three or four times, when a few taps on the sides loosen the punch and break the chip. In order to facilitate the breaking the opening at the point is made sometimes three-sided (as in Fig. 277) instead of circular. Two men will sample a carload of bullion weighing from 17 to 20 tons

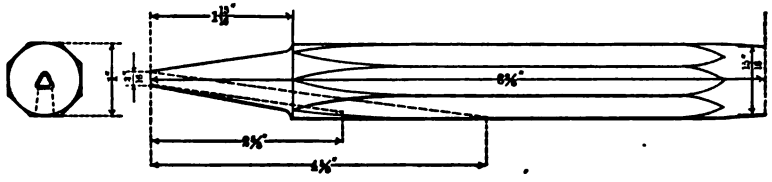


FIG. 276 AND 277.—SAMPLE PUNCH FOR BASE BULLION.

in about two hours. One serious objection to this method of sampling is that on account of the flow of metal, the hardness of the dross, and the unevenness of the surface, the chips from the top of the bar are liable to be smaller than those from the bottom. Kempf and Nenninger* tried to correct this by melting down top and bottom samples separately, then taking from the resulting

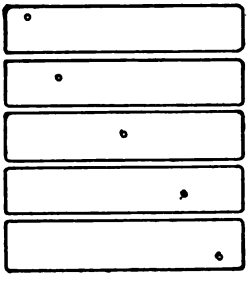


FIG. 278 —PUNCHING OF BASE BULLION.

bars equal amounts and melting them into a second bar, their final sample bar. But by the two meltings a new source of error is introduced which is liable to give more trouble than the fact that the bottom samples are slightly longer than those from the top.

Austin† recommended years ago removing from a bar 4 in.

* *Engineering and Mining Journal*, July 1, 1882. † *Ibid.*, Sept. 9, 1882.

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thick a chip a little longer than 2 in., slipping it into a hole bored 2 in. deep into a block, and trimming off the projecting end with shears. This is not followed.

At some works a drill has been introduced which removes a cylindrical chip of the desired length from the bar. The operation requires no hard work and is quick; the bit has no complicated construction and outlasts many punches; the whole apparatus is simple and light, so as to be easily moved from one part of the works to the other.

Drilling one or more holes through every bar and taking the borings as sample has been suggested, but is not used.

Sawing of bars as a means of sampling was suggested by Pattinson.* It was practiced for a considerable time by one of the Colorado smelting works. The method is as follows: The bullion is unloaded from the car into an inclined iron trough having rollers at the bottom over which it runs on to a table. Here a circular saw is so arranged with a slide and gauge as to make an incision halfway through the bar. The cuts are made respectively in the first bar at one-sixth of the bar-length, in the second at two-sixths and so on. The sample from a 20-ton lot weighing about 70 lb. is thoroughly mixed, quartered down like an ore-sample and then assayed in quantities of half an assay ton without any previous melting, as results obtained after smelting are liable to be too low. They are said to have agreed very well with the usual punch-samples taken diagonally across the bars, and also with the output of the refinery. Raht† objects to this and all methods of sampling bullion in the solid form. He melts down the bullion in a kettle, drosses and stirs it for five minutes, and then takes a number of dip-samples, each weighing about ½ assay ton, either from the kettle or from the siphon while the bullion is being drawn off into molds. These samples are to be weighed separately and assayed and the results figured as ounces per ton. This method of sampling has been adopted at a number of smelting works, but not the weighing out of the whole sample and calculating the result.

The chips obtained by sampling a carload (weighing from 17 to 20 tons) or a lot are collected in a wooden box, melted down,

* *Journal Society Chemical Industry*, 1892, xi., p. 321.

† "The Mineral Industry," iii., p. 417.

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and cast into a bar. The writer* has always obtained satisfactory results by the following procedure: The chips are melted down quickly in a graphite crucible, which has first been heated for some time, so as to be dark-red near the bottom. When melted and heated till the dross is about redissolved, but before any cupelling has begun, the lead is well stirred with an iron rod

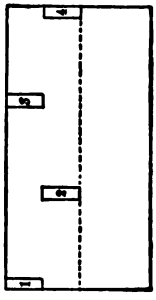


FIG. 279.—SAMPLE BAR.

for several minutes and then poured *with* the dross into a mold of, say, 3 by 7 in. This sample-bar, representing from 17 to 20 tons of base bullion, weighs from 8 to 10 lb. Four samples are taken from the middle of the sides of it, cutting the entire bar through. Another method of melting is to bring a No. 10 graphite crucible to a white heat, then take it out of the furnace and drop the samples into it. They will melt down quickly. Stir and pour into a mold 2 in. thick which will give a bar 9 by $3\frac{3}{4}$ by $\frac{1}{4}$ in., weighing 6 lb., or one 10 by 5 by $\frac{1}{2}$ in., weighing 10 lb., as represented in Fig. 279. The bar is cut in two along the dotted line; one half is reserved for the umpire assay; from the other half the four samples (Nos. 1 to 4), each weighing a little over $\frac{1}{4}$ assay ton, are cut out to be assayed. It is essential that the iron mold should be cold and thick so that the bar when cast shall solidify quickly and the precious metals have no chance to become unevenly distributed. At the Delaware Lead Works the samples were melted down at a very low temperature (some sawdust being added to make the rising dross as dry as possible), the lead was poured into a mold and the dross raked out on an iron plate and

* "Mineral Resources of the United States," 1883-84, p. 464.

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rubbed. Lead and dross were weighed and assayed and the result calculated as follows:

$$\frac{A.a+B.b}{A+B.}=\text{Average assay.}$$

A.=weight of dross; a=assay of dross.

B.=weight of lead; b=assay of lead.

A number of such weights and assays are given above. Where a large quantity of base bullion is desilverized, the method takes too much time. With bullion rich in dross it is to be recommended.

§ 84. SPEISE.—The speise obtained in lead-smelting is principally an arsenical speise. Antimonial speise is occasionally made in smelting softening skimmings (§ 128). The predominant element of arsenical speise is iron which is partially replaced by nickel, cobalt, copper, and to a small extent lead, bismuth, gold, and silver.

	Leadville.	Leadville.	Pueblo.	Pueblo.	Denver.
Ag	0.0085	0.0301	up to 0.016	Up to 0.014	35.00h
Au	Trace	0.0009			0.05h
Cu	0.3628	0.2566	2.09	5.06	0.95
Pb	1.4935	2.5080	1.87	0.69	
Mo	0.2110	0.2155			
Fe	60.5780	70.4780	58.33	59.42	59.10
Zn	Trace	Trace			
Ni	0.0876	0.0981	Trace	Trace	
Co			Trace	Trace	
S	5.8191	4.4695	4.105	2.80	6.88
As	31.4725	21.8003	30.005	31.17	31.54
Sb	Trace	0.1450	Trace	Trace	
SiO_2			0.15	Trace	
CaO					
Reference	(a)	(a)	(b)	(b)	(c)

	Denver.	Denver.	Eureka.	Pribram.	Pribram.	St. Andreasberg.
Ag	12.90h		0.039	0.087	0.020	
Au	0.10h					
Cu	4.00	5.25	1.06	1.956	0.409	18.37
Pb	2.00	2.01	2.18	1.752	3.245	66.84
Mo			2.31			
Fe	56.80	67.90	57.02	61.330	56.700	0.23
Zn			0.07			0.04
Ni				2.056	0.763	0.13
Co				0.194		
S		4.34	3.34	9.600	10.000	Bi Trace
As	32 50	29.80	32.95	18.750	26.757	Trace
Sb		0.30	0.13	2.450	1.608	3.60
SiO_2			0.23			Sn 10.60
CaO			0.34	CaO 0.500	CaO 0.500	
Reference	(c)	(d)	(e)	(f)	(f)	(g)

(a) Emmons, "Geology and Mining Industry of Leadville," p. 730. (b) Dewey, Bulletin No. 42, "United States National Museum," p. 52. (c) Curtis, "Silver-Lead Deposits of Eureka, Nev.," Monograph vii., "United States Geological Survey," 1884, p. 160. (d) Balling, *Berg- und Hüttenmännische Zeitung*, 1867, p. 419. (e) Private communication of M. W. Iles. (f) *Ibid.*, C. H. Livingstone. (g) Hampe, *Berg- und Hüttenmännische Zeitung* 1893, p. 140. (h) Ounces per ton.

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Guyard calls attention to the absence of cobalt in Leadville speise, which he found concentrated in the dross skimmed from the lead-well. He also found as much as 10% of grains of metallic iron free from arsenic in Leadville speise, which is uncommon. It has been already stated that speise always contains shots of lead, and that when coarsely crystalline less is found than when it is fine-grained. As regards the presence of precious metals, the fact is to be noted that speise retains considerable

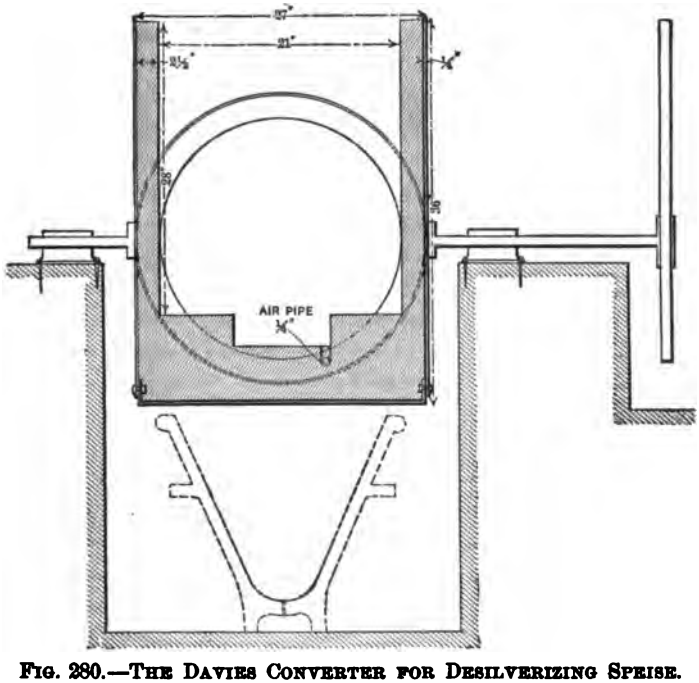


FIG. 280.—THE DAVIES CONVERTER FOR DESILVERIZING SPEISE.

amounts of gold, while very little gold is found in matte; speise-assays show from a trace to 0.5 oz. gold per ton.

To treat speise so as to extract the silver, gold, and copper economically has always been a difficult problem. With large quantities the cheapest way is to roast it in a heap of about 50 tons, which burns from two to four weeks. The imperfectly roasted speise is sorted out, crushed and roasted in a calcining furnace. The whole is then smelted in the blast furnace with pyrite or matte. The result will be base bullion and a matte

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rich in copper and silver, and perhaps a small amount of speise, in which any nickel and cobalt will be concentrated. This second speise goes to a new heap of first speise, as nickel and cobalt occur in such small quantities as not to call for any further attention.

With the small amount formed to-day, the simplest way is to crush it and roast it with sulphurets in the proportion of 1 : 10, in the reverberatory furnace, when the sulphur trioxide set free will decompose the arsenides and arsenates, converting them into sulphates.

Davies* invented a process for desilverizing speise which is said to give satisfaction at the Eureka Consolidated Works, Eureka, Nev. It consists in tapping 800 lb. of speise into a small cylindrical iron converter lined with firebrick (Fig. 280), adding from 20 to 25% liquid lead and introducing from the bottom a blast of 17 oz. pressure through a ¼-in. pipe for three or four minutes. This stirs up the lead and speise and burns off some arsenic. Most of the silver and gold is taken up by the lead; the liberated iron corrodes somewhat the lining, but eats through only very slowly. The converter is turned down and the contents are discharged into a cast-iron receiver having the form of a slag-pot, the bottom of which has a ½-in. hole for tapping the lead. The desilverized speise solidifies quickly, and when hard the still liquid lead is tapped. Best results are obtained when the lead assays 40 oz. to the ton. The desilverized speise is then a waste product. The claim is made that from a speise of the composition shown in the table, 85.5 of the silver and 89.28 of the gold are extracted, with a loss of lead varying from 5 to 8%.

§ 85. MATTE.—The matte produced in lead smelting is principally a compound of iron and sulphur, in which the iron has been in part replaced by lead and copper, and in a less degree by zinc, silver, nickel, cobalt, manganese, arsenic, antimony, calcium, barium, and magnesium.

In the two analyses by Hampe of matte made at Clausthal and in the analysis by Mann of matte made at Pribram, the presence of oxygen is to be noted. Hampe believes that the oxidation of the matte occurs in tapping. The presence of magnetic oxide of iron in Montana copper mattes was proved by Keller.†

* *Engineering and Mining Journal*, June 30, 1888.

† *Engineering and Mining Journal*, Nov. 16, 1895; see also Keller, "Transactions of American Institute of Mining Engineers," xxii., p. 580, and Neill, *Ibid.*, p. 675.

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The very high percentage of lead in the matte produced formerly (1867) in the Upper Harz (analysis *a*) was connected with an enlargement of the furnace at the zone of fusion, made to lower the temperature and thus reduce the volatilization of lead and the corrosion of furnace walls. This resulted, however, in an incom-

	Clausthal, Old. (*a*)	Pribram. (*b*)	Pueblo. (*c*)	Clausthal. (*d*)		Pueblo. (*e*)	
Pb	41.50	11.16	10.72	15.500	12.44	16.00	15.95
Cu	0.86	1.59	0.61	6.295	3.49	43.50	58.51
As	.....	0.55	0.56	.....	.....	0.14	0.10
Sb	0.66	0.98	None.	0.120	0.13	0.02	Trace.
Sn	.....	Trace.	.....	.....	.....	0.175	0.066
Ag	0.12	0.105	0.084	0.004	0.035	.....	.....
Fe	34.05	41.31	52.27	42.940	48.64	12.30	6.05
Zn	.....	11.55	4.27	6.000	4.31	3.60	1.24
Mn	.....	.....	.....	Trace.	Trace.	1.20	0.42
MnO	.....	1.40	.....	K_2O 0.170	0.13	.....	.....
Ni	.....	Trace.	.....	0.260	0.11	.....	.....
Co	.....	Trace.	.....	Na_2O 0.280	0.14	.....	.....
SiO_2	.....	3.06	.....	1.560	1.22	2.20	.....
Ca	.....	.....	.....	SO_3 0.220	n.d.	.....	.....
CaO	.....	0.05	0.41	0.156	0.51	.....	.....
Mg	.....	Trace.	0.47	MgO 0.294	MgO 0.05	.....	.....
Al_2O_3	.....	Trace.	.....	0.770	1.51	.....	.....
S	23.82	22.23	24.015	23.921	25.65	19.20	17.70
O	.....	4.79	.....	1.500	2.08	.....	.....

	Pueblo. (*e*)		Denver. (*f*)				Leadville. (*g*)
Pb	8.50	12.45	8.0	9.0	10.0	14.8	18.0
Cu	20.40	10.35	3.0	4.7	8.0	9.5	10.4
As	Trace.	0.12	.....	.....	.....	.....	.....
Sb	Trace.	0.05	.....	.....	.....	.....	.....
Sn	0.16	0.213	Au 0.05*h*	Au 0.02*h*	Au 0.07*h*	Au 0.09*h*	Au 0.02*h*
Ag	.....	.....	51.7*h*	76.0*h*	56.9*h*	95.7*h*	106.8*h*
Fe	41.20	42.50	37.5	46.2	38.4	31.6	47.6
Zn	1.50	3.10	6.0	.....	8.5	.....	.....
Mn	1.40	3.30	.....	.....	.....	.....	.....
MnO	.....	.....	.....	.....	.....	.....	.....
Ni	.....	.....	.....	.....	.....	.....	.....
Co	.....	.....	.....	.....	.....	.....	.....
SiO_2	0.20	0.30	7.8	1.4	3.3	4.2	1.2
Ca	.....	.....	.....	.....	.....	.....	.....
CaO	.....	0.15	.....	BaO 0.46	.....	.....	.....
Mg	.....	.....	.....	.....	.....	.....	.....
Al_2O_3	.....	.....	.....	.....	.....	.....	.....
S	26.30	26.40	28.0	22.6	.....	20.4	23.0
O	.....	.....	.....	.....	.....	.....	.....

(*a*) Balling, "Metallhüttenkunde," p. 86. (*b*) Mann, *Oesterreichisches Jahrbuch*, xxxix., p. 24. (*c*) Dewey, Bulletin No. 42, "United States National Museum," p. 52. (*d*) Hampe, *Chemiker Zeitung*, 1892, xvi., p. 458. (*e*) Private communication of W. H. Howard. (*f*) *Ibid.*, M. W. Iles. (*g*) *Ibid.*, R. D. Rhodes. (*h*) Ounces per ton.

plete decomposition of galena. In modern furnaces that are contracted at the smelting zone a matte contains from 8 to 12% of lead, if not otherwise enriched by the presence of zinc in the charge. It often happens that, if the constituents of a matte are figured as sulphides, the analysis does not show enough sulphur to combine with the metals as FeS, PbS, Cu_2S, NiS, etc. The

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explanation given by Rammelsberg* for lead sulphide, and by Münster,† Schweder,‡ and Farbaki§ for iron and nickel sulphide, is that, as subsulphides do not exist, lead, iron, and nickel are held in solution by their sulphides while liquid, and separate out during solidification. Mackintosh‖ claims the existence of a subsulphide of nickel and iron. The value of a matte depends on the amount of silver, gold, lead, and copper, and (in exceptional cases) of nickel and cobalt it contains. Matte contains considerable silver, but not much gold. The silver increases with the lead and this as a rule with the copper. It was formerly considered satisfactory if the first matte assayed one-third as high in silver as the base bullion. Now, except under peculiar circumstances it assays one-fifth as high. In concentrating matte by itself 50% of the silver ought to be collected in the bullion produced. The tendency of gold is to enter almost entirely the base bullion. This is illustrated in a very interesting way by the fact observed by Carpenter,¶ that if a gold and silver-bearing iron matte produced in pyritic smelting be added raw to the charge of a lead blast furnace, all the gold, and from 60 to 70% of the silver, will be taken out. The first matte produced in a lead furnace rarely contains more than 5% copper and from 1 to 3% nickel-cobalt.

The object aimed at in working the matte is to extract the silver and gold by means of lead, and to concentrate the copper into an enriched intermediary product, which is either sold to copper works or brought forward to metallic copper to be desilverized by electrolytic refining or by vitriolization. The two operations by which matte is concentrated consist of roasting and smelting, with an acid flux to slag the iron and a lead ore to take up the silver and gold. Bessemerizing** of lead-bearing matte is connected with great loss in silver, thus in blowing matte with 10% lead up to blister copper a loss of 33% silver took place.

§ 86. Roasting of Matte.—Roasting is carried on in heaps, stalls, kilns, and reverberatory furnaces. The choice of method

* "Lehrbuch der Chemischen Metallurgie," Berlin, 1865, pp. 48 and 232.
† *Berg- und Hüttenmännische Zeitung*, 1877, p. 195.
‡ *Ibid.*, 1879, p. 18; *Iron*, xiii., p. 292.
§ *Berg- und Hüttenmännische Zeitung*, 1894, p. 184.
‖ "Transactions of American Institute of Mining Engineers," xvi., p. 117.
¶ Private communication, May, 1892.
** Peters, "The Mineral Industry," iii., p. 230.

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will depend greatly on the richness of the matte in silver. In roasted matte, according to Plattner,* the exterior part of each piece or pellet contains more silver and lead than the interior. Any dust that is made in handling the roasted matte, coming from the surface, will cause considerable loss in silver and lead; again, if exposed to the rain, silver sulphate formed in roasting is liable to be lost by leaching; finally, with rich matte the aim must be to roast as quickly as possible, so as not to lock up too much capital for months at a time.

1. *Roasting in Heaps* has the advantage of cheapness and simplicity of plant; the matte need not be crushed fine, and most of the roasted matte is obtained in lump form, which is desirable for the blast furnace. This is, however, more than made up for by the many disadvantages, as loss of metal by dusting and leaching, the consumption of fuel, slowness of roast, imperfect roasting, with consequent necessity of re-handling and re-roasting, and obnoxious fumes. Therefore heap-roasting has not found much favor for matte. Where small quantities of matte are produced and where a calciner does not exist, heap-roasting is in place.

A heap for roasting matte should not be very high, as the fire then becomes too hot and melts the central part of the heap; on the other hand, if very low, the roast will be imperfect and the loss in silver and lead large. The sides of the heap are always covered with smalls, to prevent the fumes from passing off there. The height of a matte-heap varies from 5½ to 7 ft.; the length and width have not much influence on the result.

The roast-heap is usually built on the level of the feed-floor, as the fumes will not enter the smelter building, and it is important not to carry the roasted matte any further than necessary. The loss incurred in transporting the raw matte to the roast-heap ought not to be worth taking into account. The ground for a heap should be even, hard, and slightly elevated. Soft ground is leveled by a plow or a scraper; hard ground is made even by filling; the elevation of 6 in. is obtained by making a bed of coarse slag, covering it with fine slag, and rolling it, if possible. The first matte, being rich in iron, oxidizes and crumbles when exposed to the air for a short time, so that the breaking-up into pieces of fist-size does not cause much labor. In building the

* "Die Metallurgischen Röstprocesse," Freiberg, 1856, p. 205.

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heap, a bed of light wood from 6 to 15 in. deep is made, leaving channels (10 in. wide) open every 6 ft., which are filled with kindling. Hard wood is not so good for the bed, as it makes too hot a fire, which is liable to fuse the lower part of the heap. Sometimes a chimney is placed in the center, consisting of four boards nailed together and long enough to reach 2 ft. above the heap. It helps to start the heap uniformly. In building the heap, first coarse matte is piled around the chimney, and then distributed over the bed of wood. The coarse matte is assorted from the finer matte with a sluice-fork. When the heap has been built in the form of a pyramid, the edges of which reach to within a ½ ft. of the border of the wood, the sides are covered with fine matte and sometimes also the top. Fine matte is, however, kept on hand, to be used when the heap has been started, to make the top layer thicker, and to check the draft in parts of the heap that are becoming too hot. In Utah* the pyramidal heaps are made 24 by 18 ft. at the base and 6 ft. high; they contain about 80 tons of matte, and burn from 30 to 40 days.

In the Harz Mountains† matte-heaps are usually made 15 by 23 ft. on a bed of wood 6 in. deep; they contain from 75 to 100 tons and burn 30 days. Quite a saving of cordwood is effected by placing pieces of wood, split from sound dry logs about 9 in. in diameter, end to end in parallel rows, about 12 in. apart, on the ground, and laying small sticks across them. On this bed are distributed in rows above the cordwood large pieces of matte to the height of about 9 in. Between these are piled fagots and soft coal. Looking at the heap from the top there will be seen alternate rows of coarse matte and fagots mixed with coal. On this bed the heap is built in the usual way.

The well-roasted part of a matte-heap amounts to about 33%, if the roasting has been carefully conducted, and the sulphur has been reduced to about 6%. Analyses‡ of raw and roasted matte from Clausthal are subjoined:

Matte.	Pb.	Cu.	Sb.	Ag.	Fe.	Zn.	Mn.	Co.	Ni.	SiO_2	CaO	MgO	S.	O.	SO_3.
Raw....	10.665	4.620	0.267	0.0299	53.112	2.110	0.385	0.315	0.097	0.510	0.383	0.054	26.877	.....	.....
Roast'd	10.492	4.123	0.128	0.0327	52.411	2.459	0.317	0.239	0.111	1.486	0.336	0.061	0.613	22.9053	4.225

* Terhune, "Transactions of American Institute of Mining Engineers," xvi., p. 23.

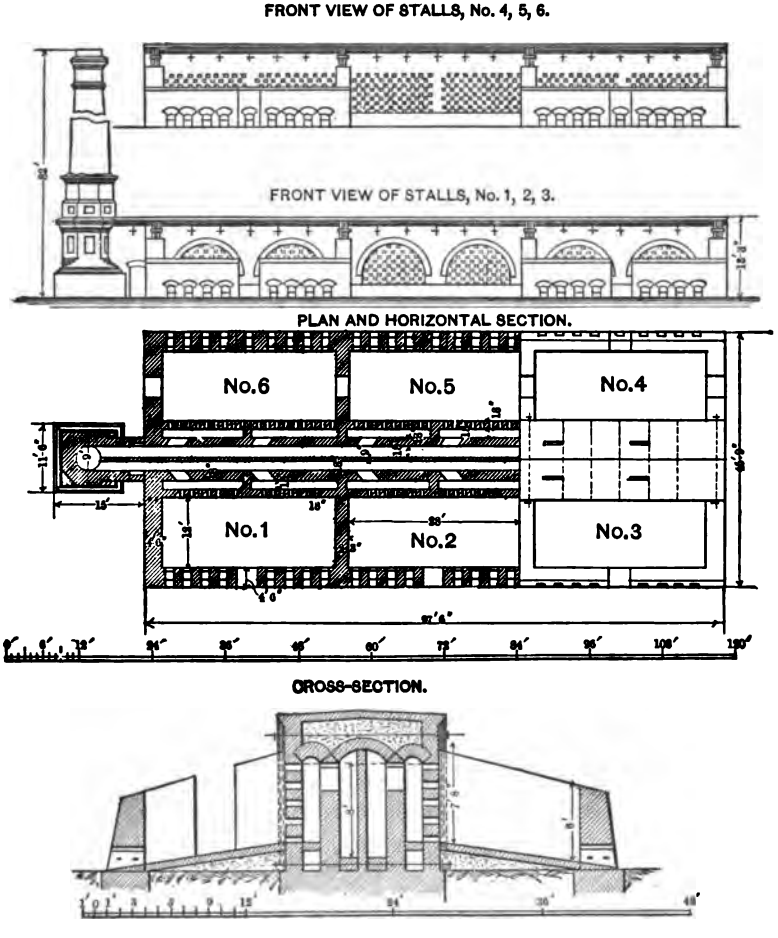
† Private notes, 1890.

‡ Arche, "Die Gewinnung der Metalle," Leipsic, 1888, p. 72.

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To roast one ton of matte in 80 or 100-ton heaps, about 0.3 cords of wood are required and from 5 to 8 days labor.

2. *Roasting in Stalls* has many advantages over roasting in



FIGS. 281 TO 284.—PRIBRAM ROASTING STALLS.

heaps. There is less loss of metal by dusting and none at all by leaching; little fuel is needed; the piles may be smaller; the roast is quicker and more perfect, requiring less matte to be

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roasted a second time, and the fumes are carried off. The only drawback is the cost of plant, and this is more than made up, especially if the stalls are built of slag-brick, by the advantages. That stalls have not superseded heaps is because, if a plant has to be built, it is considered that a long-hearth reverberatory furnace may as well finish the roasting by one short operation. The writer, however, does not agree with this view, and thinks that stalls might in many cases be preferable.

The open stalls used at Pribram* (Figs. 281 to 284) will serve as an illustration. The general arrangement is shown by Figs. 281 and 282. There is a double row of stalls placed back to back with a flue between them, which is divided by a vertical wall into two parts, in order that each row of stalls may have an independent draft. The flue terminates in a chimney 82 ft. high. The side walls of a stall are built up solid; in the front wall on each side of the entrance are four fireplaces, from which the heap is kindled. The small flues in the back wall open into two chambers, which connect separately with the main flue, and can be closed off by sliding dampers. The advantage of the divided flue is that in treating small quantities of matte one side only of the stall is filled, and when the roast is finished the clinkered and insufficiently roasted parts can be placed directly on the heap which is being erected on the other side. Stalls 1, 2, and 3 receive small charges of 65 tons, and stalls 4, 5, and 6 large charges of 140 tons. The stalls are built entirely of slag-brick.† They are made by tapping the blast-furnace slag from a Devereux slag-pot and letting it run into a cast-iron mold. The upper surface is made smooth by placing a heavy cast-iron plate on the mold when filled. If the slag is not sufficiently acid, it cannot be used alone for this purpose, as it cracks while cooling. It must then be used to cement together clinkers, sharp-edged sand, pieces of brick, etc., placed in the mold. It is well to cover the mold with sand or ashes when filled, in order that the slag may cool slowly, as it is less liable to crack. Slag-bricks form an excellent building material for the stalls. They are laid with

* Zdráhal, *Oesterreichisches Jahrbuch*, xxxix., p. 16, and private notes, 1890.

† For manufacture of slag brick see: Peters, "Modern Copper Smelting," New York, 1895, p. 142; Egleston, *School of Mines Quarterly*, xii., p. 189; Klette, *Zeitschrift für Berg-Hütten- und Salinen-Wesen in Pruessen*, xl., p. 501, and *Berg- und Hüttenmännische Zeitung*, 1893, p. 150.

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ordinary clay mortar; the walls are sufficiently thick not to require any binding.

Four sizes are made:

No.	Length. Inches.	Width. Inches.	Height. Inches.
1	17¼	8¼	8¼
2	17¼	4⅛	8¼
3	12½	12½	6¼
4	12½	6¼	6¼

Bricks Nos. 1 and 2 are used for the side walls; Nos. 3 and 4 for the bottom of the stall. Nos. 2 and 4, having half the width of Nos. 1 and 3, correspond to split brick, and are used as binders.

Three improvements in construction for the admission of air may be mentioned. The first consists in leaving open two horizontal air-passages, one above the other, in each side wall, with openings into the stall. One may be placed at half the height of the side wall, terminating at half its depth; the other at three-quarters the height, ending at three-quarters the depth. The second is to leave openings in the front wall. The third is to have in the floor from 2 to 3 ft. from the back a grating about 18 in. square which is connected with an underground flue, coming to the surface some distance from the front of the stall. This introduction of air at different parts of the stall gives better control of the roast.

In charging the stall coarse matte is placed on a thin bed of wood arranged on the slag-bottom in the same way as when a heap is erected. Then follows a mixture of coarse and fine matte and the top is covered with sufficient fines to prevent the fumes from passing through the top, thus enabling them to be drawn off through the chimney. If only one-half of the stall is to be used, the open side is built up of coarse matte. The stalls are fired from the front with wood, and the fires kept going with bituminous coal and lignite until the heap is well ignited.

The time required for roasting is:

	65 Tons.	140 Tons.
	Days.	Days.
Unroasted lead matte.....	48	86
Once-roasted lead matte..	43	80
Twice-roasted lead matte..	33	70

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The well-roasted matte contains from 8 to 10% sulphur, one-third of the sulphur being present as undecomposed sulphide. It makes no difference in this result whether the stall is entirely filled or only half-filled. The change in weight after roasting is insignificant.

The labor necessary per ton of matte is:

	65 Tons.	140 Tons.
	Hours.	Hours.
For the first roast	1.81	2.02
For the second roast	2.26	2.44
For the third roast	2.26	2.44
For pulling down and wheeling to the feed-floor	3.62	3.62

With four men working in the daytime only in 10-hour shifts, the number of days required to fill and empty the stalls will be:

	65 Tons.	140 Tons.
	Days.	Days.
Filling with raw matte for first roast	3	7¼
Pulling down and filling for second roast	3¾	8¾
Pulling down and filling for third roast	3¾	8¾
Pulling down and wheeling to feed-floor	6	13
Handling	16½	37¾

The fuel required per ton of matte is:

	65 Tons.		140 Tons.	
	Wood, Cords.	Coal, Pounds.	Wood, Cords.	Coal, Pounds.
For first roast	0.008	68.2	0.007	66.0
For second roast	0.008	72.6	0.007	78.0
For third roast	0.008	71.6	0.007	84.4
Average	0.008	70.8	0.007	76.1

Stalls were formerly used at the works of the Colorado Smelting Co., Pueblo, Colo.,* when little matte was being produced. Their arrangement was similar to that shown in Fig. 148, page 210. The stalls were 15 ft. long, 10 ft. wide and 5 ft. high and had sheet-iron covers luted down. The bottoms were 2 ft. above the tracks on which the roasted matte was run to the blast furnaces. At the back of a stall were two arches filled in with a pigeon-hole wall. The main flue was 2 ft. 6 in. wide and 6 ft.

* Private notes, August, 1896.

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high and the stack 4 by 4 ft. inside and 60 ft. high. In the floor of the stall, 2½ ft. from the back, was an opening 18 in. square closed by a grating which connected with a flue under the floor ending in the side wall of the depressed track. A stall held from 40 to 45 tons raw matte and from 25 to 35 tons half-roasted matte. On the bottom was placed a layer of cordwood (piñon) 22 in. thick and matte crushed to pass a 2-in. ring piled to within 1 ft. of the top. When speise was roasted with the matte a 6-in. layer of slack coal was spread out below the top of the charge. The roasting lasted from four to six weeks, and one-third well-roasted matte with less than 2% sulphur was obtained, while two-thirds had to go to a second fire lasting four weeks, and from this one-third to a third fire also lasting four weeks. Six men (at $1.75 each) and one foreman (at $2.25) ran the plant; wood was $2.50 a cord. The cost ranged from $1 to $2 per ton, when roasting in the long-hearth hand-reverberatory furnace cost from $2.50 to $3 per ton.

An analysis* of stall-roasted ore from the Colorado Smelting Co., Pueblo, Colo., is subjoined: Fe_2O_3, 80.39%; Mn_2O_3, 0.93%; PbO, 7.91%; ZnO, 0.98%; CuO, 3.22%; As_2O_3, 0.86%; Sb, trace; S, 1.01%; SiO_2, 3.21%; CaO, 3.22%; Ag, 0.0614%.

3. *Roasting in Kilns* is practiced only in connection with the manufacture of sulphuric acid. In the Harz Mountains† and in Saxony‡ shaft furnaces are used for roasting. At the Lautenthal Works§ six Freiberg shaft furnaces,‖ each having four doors, are in use. The inside dimensions are: height, 10 ft. 6 in.; width, 4 ft. 2 in.; depth, 7 ft. 5 in. They hold about 300 cu. ft. of ore. In 24 hours one furnace roasts in two charges 4,200 lb. of a mixture of 4 parts raw first matte (4% copper, 10% lead) and 3 parts half-roasted matte, or 5,400 lb. of a mixture of 5 parts raw second matte (12% copper, 7% lead) and 4 half-roasted matte. The roasted matte contains from 7 to 8% sulphur, and the gases 5.3 volumes sulphur dioxide. The six furnaces are attended by eight men in 24 hours. Sufficient heat is generated by roasting to have a Glover tower; the sulphuric

* Dewey, Bulletin No. 42, "United States National Museum," p. 52.

† "Das Berg- und Hüttenwesen des Oberharzes," Stuttgart, 1895, p. 249.

‡ Merbach, *Berg- und Hütten-Wesen*, Freiberg, 1893, p. 307.

§ Private notes, 1890.

‖ Drawings in *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xix., plate viii.

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acid chambers have a capacity of 670,000 cu. ft., and terminate in a Gay-Lussac tower. In 1879, when the writer was at Lautenthal, there was a loss in the manufacture of sulphuric acid, and it was carried on only to make the sulphurous gases innocuous; since 1890, through the improvements in plant, acid is made with profit. Details about the management of furnaces and the manufacture of sulphuric acid are given in Lunge's excellent work.*

4. *Roasting in Reverberatory Furnaces.*—Little need be added on the roasting of matte to what has already been given under roasting of ores. From Plattner's experiments, already mentioned, which show that the silver and lead in roasted matte is concentrated toward the surface, it can be seen that the loss in these metals by dusting must be greater with revolving furnaces than with those having a stationary hearth. Up to the present time hand reverberatory furnaces have been most satisfactory in eliminating the sulphur although their capacity remains small.

Terhune† gives some interesting figures on a short experiment in roasting matte in a Stetefeldt furnace.

Before roasting, the matte must be reduced in size, which is usually done by crushing, occasionally by granulating. If it is to be crushed, the cakes are broken with sledges to sizes suitable for the rock-breaker. This discharges into rolls set to ½ in. which deliver the matte either directly into a trommel with ½-in. holes or first into the boot of a bucket elevator. When much crushing has to be done there are two pairs of rolls for roughing and finishing so as to expedite the work. The operation is usually carried on in the sampling department.

In granulating matte, the main point is to have a thin stream of it. Bretherton‡ describes the method formerly used at the American Smelter, Leadville, Colo. The plant consisted of three wooden tanks 10 ft. long, 5 ft. wide and 6 ft. deep filled with water up to 10 or 12 in. from the top. These served in rotation, two being in constant use, while the third was being cleaned up. The water from the granulating tanks overflowed into a settling tank, but the amount of slime caught was small. For cleaning up, each tank had a number of openings in one side to discharge

* "Sulphuric Acid and Alkali," London, 1891, vol. i.

† "Transactions of American Institute of Mining Engineers," xvi., p. 22.

‡ *Engineering and Mining Journal*, Jan. 9, 1897.

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the water and a gate in one end to remove the granulated matte. At the head of a tank, just below the top, was inserted a 2-in. pipe, flattened out at the end to throw a thin, wide stream of water, the pressure of which ranged at from 25 to 30 lb. per sq. in. The matte was either tapped from a settling pot or poured from an ordinary slag-pot. In the latter case iron bars were placed over the granulating tank to prevent the shell from falling into it and causing violent explosions. The thin stream of matte striking the water issuing from the nozzle was scattered into droplets and chilled before it fell into the water.

Another plant working on the same principle under a pressure of 80 lb. has two granulating tanks, 12 ft. long, 4½ ft. wide and 2 ft. deep, discharging into one settling tank 12 ft. long, 9 ft. wide and 2 ft. deep, which has near the discharge end a canvas partition to prevent any float-matte from being carried off.

The J. MacArthur granulating trough (page 272) also does most satisfactory work with matte and makes no float-matte.

Granules from lead-bearing matte are said to be hollow and to decrepitate upon heating in the reverberatory furnace. With low-grade copper matte containing from 10 to 15% copper this does not seem to be the case,* it being difficult to roast the granules satisfactorily; granules of 30 or 40% matte give up their sulphur more readily.

§ 87. SMELTING OF ROASTED MATTE.—The first matte obtained in ore smelting (containing about 7% copper) is, after being roasted, always added as iron flux to the ore-charge in the blast furnace. The resulting second matte, containing from 10 to 13% copper, is concentrated in the blast furnace, in the reverberatory furnace, or in the two successively.

1. In the Harz Mountains the matte is concentrated exclusively in the blast furnace to black copper (76% copper) and an enriched copper matte (67% copper). As many as four separate heap-roastings, each followed by a reducing smelting in the blast furnace, are carried on to obtain the final products that are sold to copper works.

The roasted matte is smelted with 147% slag and 15% coke in a low blast furnace having a crucible that is partly internal and partly external, or only external.

One hundred parts of roasted second matte give 13.06 parts of

* Peters, "Modern Copper Smelting," New York, 1895, p. 97.

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impure base bullion and 35.42 parts of third matte. From this on no more lead can be extracted. The rate at which the matte decreases in quantity as its percentage of copper increases is shown by the following tables:

AMOUNT OF MATTE.

First matte	60.27
Second matte	21.00
Third matte	10.96
Fourth matte	4.57
Fifth matte	3.20
Total	100.00

ANALYSES OF MATTE. (*a*)

	Matte I.	Matte II.	Matte III.	Matte IV.	Matte V.
Pb	13.47	8.33	10.02	9.06	3.88
Ag	0.035	0.040	0.065	0.075	0.085
Cu	5.73	12.86	27.75	42.30	59.86
Fe	48.18	44.60	31.25	19.98	12.24
Zn	1.71	3.32	2.99	1.59	0.75
S	25.25	20.42	21.92	17.89	20.78

ANALYSES OF SLAG. (*a*)

SiO_2	23.73	25.30	27.43	17.43	18.97
$BaSO_4$	.....	0.36	1.06	Trace	.....
Pb	1.68	1.53	1.42	1.47	0.92
Cu	0.58	0.63	0.56	1.90	1.58
Ag	0.002	0.0019	0.0013	0.0015	.....
Sb	0.14	0.05	0.14	0.09	0.20
FeO	48.16	51.99	49.67	62.14	61.84
Al_2O_3	8.12	3.86	5.28	5.87	6.47
P_2O_5	0.41	0.56	1.31	0.43	0.47
MnO	0.60	0.71	0.77	0.49	0.50
ZnO	8.11	6.69	6.38	4.30	2.90
Ni+Co	0.09	0.08	0.12	0.27	0.54
CaO	3.60	4.61	4.16	2.00	2.27
MgO	0.32	0.40	0.52	0.39	0.47
BaO	0.41	.....	.....	.....	.....
K_2O	0.42	0.47	0.64	0.57	0.77
Na_2O	0.36	0.33	0.38	0.33	0.53
S	3.95	3.49	2.39	2.64	1.48

(*a*) Private notes, 1890.

From the above analyses can be seen that the roasting of the second matte is imperfect; the results after smelting show an increase of 15% copper instead of 30%, as might have been expected. Thus the fourth matte is only as high as the third should have been. While the amount of iron in matte V. has been somewhat reduced in comparison with that of matte IV., it is still very high, and can be removed only by smelting in the reverberatory furnace. It would seem, therefore, advisable to stop the concentration in the blast furnace at 40 or 50% copper, and to treat the resulting matte in the reverberatory furnace.

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2. The general practice of American smelters is to add the roasted first matte as iron flux to the ore charge and to treat the resulting second matte, after roasting, with blast-furnace slag either in a lead furnace or in a copper furnace, if there is enough matte to warrant its erection. With a lead furnace used for this purpose, having in the side opposite the well a water-cooled bullion-tap 6 in. below the top of the crucible and in the front a tapping-jacket with two openings for matte and slag, when the change from ore-charge to matte-charge is to be made, the lead in the crucible is dipped out through the well until the lower end of the channel is laid bare. This is then closed up by tamping with clay, and ore is run until the crucible is about filled again, when the matte-charge is put on. The lead reduced from the matte is removed at intervals through the bullion-tap and matte and slag through the separate tap-holes in the breast. When the matte has been worked up, ore-charge is again put on the furnace, with extra lead if necessary. As soon as the lead begins to come quickly, the tamping in the siphon-tap is removed and the furnace run again in the usual way. During the matte-run some copper of course enters the lead, but being kept sufficiently hot it does not solidify and smelting goes on satisfactorily on a bath of mushy lead. In the course of one or two days after the ore-charge has been put on again, the coppery lead will have been replaced by fresh lead reduced from the ore-charge.

At some works the lead-tap is laid 8 in. above the bottom of the crucible, which is 24 in. deep. The method of working differs from that with the old furnaces (§ 70) in tapping only the lead from the bottom, the matte being discharged from the breast like slag.

In large works a furnace similar to a copper-matte furnace is frequently found. Often the crucible of the lead furnace is simply filled with well-beaten brasque, thus changing it to a matte furnace.

At some works the tapping-jacket has two tap-holes, the lower one for lead and matte, the upper one for slag. The pot containing lead and matte with some slag is allowed to cool on the dump. When cold the button of lead is separated from the matte, and this from the slag, the latter being smelted over again. The slag from the upper tap-hole is also allowed to cool

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on the dump, and is examined for shots of matte before it is discarded.

Again, the entire contents of the furnace are tapped into an overflow-pot. This is to collect the lead, while matte and slag overflow into a catch-pot having a tap-hole. All the slag remaining in the second pot is smelted over again. The overflow-pot has a tap-hole placed on the side, as near the bottom as possible. Every six or twelve hours, according to the grade of matte that is being smelted, this pot is wheeled over a cast-iron basin let into the ground, and the lead tapped. As soon as matte appears the tap-hole is closed with a clay plug. Other settling-pots and forehearths have already been discussed under § 70.

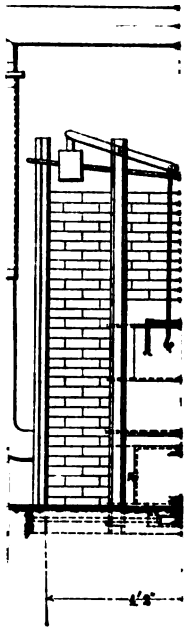
A third method of obtaining a good separation of slag and matte is to have the bottom of the furnace inclined. Thus the distance between the center of the tuyere and the slag-tap is 10 in., the latter being 8 in. higher than the matte-tap placed at the opposite end of the furnace. An excellent separation of matte and slag is effected.

The roasted matte is usually smelted with foul slag from the ore furnace and silicious copper ore low in silver and often high in gold, if this be available. The reason that the roasted matte is not smelted with dry silver ores is that only 95% of silver contained in the matte is paid for by copper refiners. The tendency is therefore not to enrich, but rather to reduce the silver contents of the matte. The slags made are similar to ore-furnace slags, with this difference, that they usually run lower in lime and are not quite so silicious. In order to reduce the lead, it is important to use a high percentage of fuel, and scrap iron forms a valuable addition to the charge. A few special slags may be quoted:

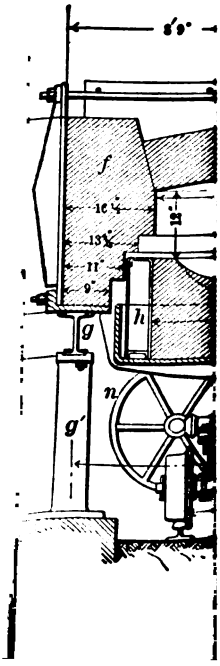
SiO_2.	FeO.	CaO.
32	36	14
32	40	16
33	58	9
30	40	10

They will show less than ½ oz. in silver per ton and from 1 to 1.5% lead. In blowing out a matting furnace it is important to collect every scrap of lead in the crucible, as it is liable to be very rich in precious metal. For example, in making matte with

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30% copper and 600 oz. silver and 2 oz. gold per ton, the lead soaked into the crucible assayed 3,447 oz. silver and 222 oz. gold per ton.

A furnace 36 by 108 in. at the tuyeres and 12 ft. high will put through, with from 10 to 12% coke, 110 tons of matte charge. The first matte obtained in the blast furnace runs from 24 to 30% copper; when this is smelted again, after having been roasted in a calcining furnace, its copper contents range from 40 to 60% when it goes to copper works.

Here it is treated by one of two methods: the first is to convert the matte into metallic copper and refine it, after which it is desilverized by electrolysis or vitriolization; the second is to dead-roast it, if it is sufficiently free from iron as at Freiberg, and then treat it with sulphuric acid to dissolve the copper, which is sold as blue vitriol, the residue going to a lead blast furnace.

Instead of shipping the 60% matte to copper works, some of the lead smelting and refining works have undertaken to work it into finished products. For instance, the Pueblo Smelting and Refining Co. worked the J. J. Crooke process* and the Consolidated Kansas City Smelting and Refining Co., the Hunt and Douglas Process No. 2,† but both processes have been given up. The common practice to-day is to roast some of the blast-furnace matte and smelt it with raw matte and a small amount of silicious flux in a reverberatory furnace, bringing it to 70% matte, which is then either roast-smelted for blister copper or part of it is roasted separately and then smelted with unroasted matte, no flux being used. The furnace is either a Welsh copper smelting furnace or it is built into an iron pan supported in a way similar to that of the English cupelling furnace. A Welsh copper smelting furnace‡ with a hearth from 10 to 12 ft. wide, 18 ft. long, and from 12 to 15 in. deep will treat a 20-ton charge in 24 hours.

A furnace of the second class of comparatively small dimensions is shown in Figs. 285 to 289. It is a combination of the

* Patent No. 308,031, Nov. 11, 1884, and No. 354,182, Dec. 14, 1886: first edition of this book p. 267.

† U. S. Patent, No. 227,902, May 25, 1880; Hunt, "Transactions of American Institute of Mining Engineers," x., p. 19; xvi., p. 80; Douglas, "Mineral Resources of the United States," 1882, p. 279; Howe, "Production of Gold and Silver in the United States," Report of the Director of the Mint, 1883, p. 796; Hunt and Douglas, *Engineering and Mining Journal*, Oct. 8, 1885; Fulton, *Ibid.*, Dec. 12, 1885; Howe, *Ibid.*, Dec. 19, 1885; first edition of this book p. 268.

‡ For details of construction see Peters, "Modern Copper Smelting," New York, 1895.

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English cupelling furnace with its fixed roof *a* and movable hearth *b* and the reverberatory furnace for concentrating matte, the flame passing from the fire-box *c* along the major axis of the hearth to a flue *d* placed over the working-door *e*. A new feature lately introduced in several cupelling furnaces is that the side walls *f* rest on I-beams *g*, supported by pillars *g'*, thus leaving the entire bottom of the hearth and test-carriage exposed to the cooling action of the air. The hearth, 6 by 8 ft., is inclosed by water jackets *h* and *h'*, and is supported by four screws *i*, with threaded sockets *j* in the test-carriage *k* running on rails. The two front screws can be rotated together by a pair of worm-wheels *l* on a shaft *m* revolved by bevel-wheel and pinion, and these by the hand-wheel *n*; in a similar way the two back-screws can be raised and lowered from the side by turning a crank (not shown). The steel water jacket *h* receives its cooling water at the back (not shown), which is drawn off at the front from the top through pipes *o* and *o'*; the cast-iron breast-jacket *h'* receives its water at *p* and discharges through two pipes *q* and *q'*; thus the jackets are always filled with water. Rubber hose connects the jackets with the supply and discharge pipes. The hearth material is a mixture of raw and burnt clay or raw clay and quartz. A chromite lining* is said to be doing satisfactory work.

The fire-box has a shaking grate, worked with underwind. On either side of the bridge is a tuyere-pipe through which blast enters upon the hearth. The furnace is incased in iron plates and is strongly bound with iron rods. It will bring forward in three days about 10 tons of 40% copper matte produced in the blast furnace and in smelting softening furnace dross and skimmings (§ 128)—to blister copper, the matte being charged in four portions, first 45%, then 25%, 22%, and 8%. The 10 tons of matte produced 8,700 lb. blister copper with 95% copper and 8,300 lb. of slag assaying 13.8% copper and 52.3% lead, and required 900 gal. of reduced oil with air as atomizer. The furnace would also be well adapted for concentrating retort bullion if the hearth were made thinner and shallower.

3. At Freiberg† the 50% matte‡ is roasted in a calcining

* Lang, *Engineering and Mining Journal*, Feb. 27, 1897.

† Grand, *Annales des Mines*, 1875, vii., p. 314; Capacci, *Revue Universelle des Mines*, 1881, ix., p. 269.

‡ Schertel, *Berg- und Hüttenmännische Zeitung*, 1888, p. 442.

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furnace and smelted in a reverberatory furnace with barite and quartz as fluxes. The hearth of the furnace is 13 ft. long, 4 ft. 2 in. wide at the bridge, 8 ft. at the middle, and 1 ft. 2 in. at the flue; the grate, 4 ft. 3 in. by 3 ft. 4 in., is 1 ft. 3 in. below the top of the bridge, which is 1 ft. 3 in. below the roof. The furnace has the usual two doors, one at the side and one beneath the flue. The bottom, which is built on an iron plate, consists first of a full course of firebrick on which is rammed firmly a mixture of fireclay and quartz, giving it the usual disk-like form, with the lowest point at the tap-hole. The thinnest part of this layer is 3 in. thick. The working-bottom, which follows, consists of 5,500 lb. of an intimate mixture of 5 parts of quartz and 1 part of matte-slag. It is melted down, after being heated and patted into shape, in 12 hours, and lasts from 15 to 18 months. Five charges, each weighing 4,290 lb., and consisting of 2,640 lb. roasted matte, 440 lb. raw matte, 660 lb. barite, and 550 lb. quartz, are treated in this furnace in 24 hours, 4¼ tons of bituminous coal being consumed. From these five charges 8,250 lb. of matte is produced. The composition of matte and slag as analyzed by Schertel* is given below:

MATTE.		SLAG.	
Pb	4.85	SiO_2	30.60
Ag	0.31	PbO	14.51
Cu	78.95	CuO	4.40
Bi	0.02	FeO	8.10
Sb	0.06	(NiCo)O	0.38
As	0.18	ZnO	1.95
Fe	0.13	Al_2O_3	1.80
NiCo	0.21	BaO	28.68
Zn	....	CaO	7.70
S	18.98	MgO	0.72
		S	0.18

In order to obtain matte running so low in iron, it is necessary to produce some bottoms. These go back to the next reverberatory charge or are added to the last blast-furnace charge to be again converted into matte. The matte forms the raw material for the manufacture of blue vitriol.

This Freiberg method of using barite in the final concentration in the reverberatory furnace will hardly recommend itself, as nobody will introduce baryta into a blast-furnace slag if it can be helped, unless Aspen, Colo., ores be used. By substituting gypsum, which, as shown by Schweder (§ 69), has the same

* *Loc. cit.*

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effect, slags low in copper might be obtained, and the resulting slag, containing lime instead of baryta, would be welcome for the blast-furnace charge. Thus it would be necessary for only a small amount of the copper from the concentrated matte to pass again through the lead blast furnace and undergo the tedious process of matte concentration.

§ 88. NICKEL AND COBALT MATTE.—The behavior of nickel and cobalt in lead matte shows some points of interest. The non-argentiferous ores of Southeast Missouri (§ 20) carry small amounts of nickel and cobalt, which are concentrated in a matte by smelting in a blast furnace having a siphon-tap. At Mine La Motte*the first matte carries from 3 to 3.5% nickel-cobalt, from 0.5 to 1% copper, and from 20 to 25% lead. When this is roasted in a calcining furnace, there must remain, when it comes to be used as iron flux, from 5 to 6% sulphur in the matte, if the loss of nickel and cobalt by slagging is to be avoided. In the same way the second matte (5–6 Ni-Co; 20–30 Pb; 1–2 Cu), when roasted requires from 7 to 9% sulphur to reduce the loss by slagging; it is then smelted with silicious matter to a third matte (12–17 Ni-Co; 35–40 Pb; 3–5 Cu), the slags nevertheless assaying 1.25% nickel-cobalt and from 2 to 2.5% lead. Attempts at further concentration in a reverberatory furnace proved unsuccessful, the slags carrying 5% nickel-cobalt and 5% lead. This illustrates the difficulty of concentrating nickel-cobalt in a matte when lead is present, which apparently drives it into the slag. The nickel can then be recovered only in part and the cobalt not at all. Then it remains to introduce arsenic and form a speise. Neill used this method successfully. The experimental furnace he built was 30 in. wide at the tuyeres, and 48 in. at the charging-door; the height from tuyeres to charging-door was 6 ft. 6 in.; the crucible, 18 in. deep, was partly internal, partly external, and had a siphon-tap. He obtained a speise with 22.53% nickel-cobalt (one-third of which was cobalt,) 6.4% lead, and 4.25% copper. The matte that formed contained 3.25% nickel-cobalt, 8% lead, and 7% copper to be retreated; the slag from 0.16 to 0.32% nickel-cobalt and from 0.6 to 0.8% lead.

§ 89. SLAG.—The composition (§ 68) and the disposal (§ 76) of lead slags have already been fully discussed. There may still

* Neill, "Transactions of American Institute of Mining Engineers," xiii., p. 634.

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be mentioned the attempts made to utilize the heat of waste slag by Engelbach and Bretherton,* and Lang† for heating the blast, and by Bretherton,‡ Giroux,§ and Howell and Ashcroft‖ for raising steam.

If slags are too rich in silver to be thrown away, and resmelting is of no avail (*i.e.*, when the ore contains considerable amounts of zinc), they can be desilverized by smelting with copper-bearing pyritic ore or matte.¶ (See Hahn, § 69.) Keller** smelted slag, estimated to run 5.3 oz. silver per ton, with 13% pyritic ore containing 10% copper and 11 oz. silver per ton in an oblong blast furnace 36 by 80 in. He put through in 24 hours 113 tons of charge with 10.9% coke, a little more than half the amount then required to smelt ore charges. There resulted 5.3% matte containing 20% copper and 92.7 oz. silver per ton, showing a saving of 80% of the silver contained in the original slag.

It was formerly the custom not to allow the base bullion to run higher in silver than 300 oz. per ton; now bullion is often made that is five times as rich. The slags from such a charge will be rich in silver and are smelted over again in concentrating matte. This is one of the best means for reducing their silver contents.

In taking the sample from a freshly-drawn pot of slag the hardened surface is perforated, and a clean steel bar inserted 3 in. into the liquid slag and quickly chilled by plunging into cold water. Sometimes slag is dipped out from the pot with a clean, cold iron ladle, poured out again after a minute or two, and the thin shell of glassy slag adhering slightly to the ladle taken as sample. The former method, however, is preferable. The sample is crushed, cut down, and ground so as to pass a 120-mesh sieve. Care should be taken that the entire final sample should pass through the sieve. The part to be weighed out should be ground for a minute in an agate mortar before placing it on the tared watch-glass of the balance. This grinding is

* *Engineering and Mining Journal*, April 25, 1893.

† *Ibid.*, July 25, 1896, and *Mining and Scientific Press*, vol. lxxvii., p. 417.

‡ *Engineering and Mining Journal*, June 2, 1886.

§ *Mining and Scientific Press*, July 15, 1893.

‖ *Engineering and Mining Journal*, July 15, 1893, July 21, 1894.

¶ Brown and Phillips, "Notes on Smelting Processes at Freiberg," Freiberg, Saxony, 1886, p. 18.

** "Transactions of American Institute of Mining Engineers," xx., p. 71.

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sometimes omitted, but it may save a great deal of trouble later on.

§ 90. WALL ACCRETIONS.*—These accretions begin in lead furnaces just above the water jackets and reach up to the feed-door. They sometimes come from the galena in the charge, or from galena formed artificially during the descent of the charge which adheres to and filters into the brick walls of the furnace; but their principal origin is in the volatilization of lead, zinc, and their compounds, which takes place in the lower parts of the furnace.

	Leadville. (a)	Leadville. (a)	Leadville. (a)	Pueblo. (b)	Pueblo. (b)	Tarnowitz (sulphide). (c)	Tarnowitz (oxide). (c)	Clausthal. (d)	Denver. (e)	Denver. (f)	Denver. (g)
S	8.291	2.600	2.725	25.98	19.74	18–9	.....	14.67	10.90	.....	.....
Pb	47.491	70.631	25.953	1.48	37.48	20–32	24	82.71	h24.06	.....	.....
PbO	0.405	.....	.....	.....	.....	.....	.....	.....	.....	21.40	.....
Bi	Trace.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Ag	0.0754	0.297	0.0944	0.0188	0.0824	0.005	0.005	.....	h29.74 oz.	.....	.....
Au	Trace.	0.0007	.....	.....	.....	.....	.....	.....	h0.197 oz.	.....	.....
Cu	Trace.	.....	.....	None.	None.	.....	.....	.....	0.80	.....	Aver'ge
Zn	6.977	Trace.	53.392	45.68	38.99	30–27	60	Trace.	18.42	.....	15.00
ZnO	0.300	.....	.....	.....	.....	C. 2	.....	.....	.....	81.00	.....
Cd	Trace.	.....	.....	.....	.....	9–6	.....	.....	.....	.....	.....
Fe	6.754	0.028	.....	6.71	0.94	.....	.....	2.08	13.36	.....	.....
Fe_2O_3	1.100	3.456	1.381	.....	.....	.....	.....	.....	.....	0.40	.....
Mn_2O_3	2.887	.....	.....	0.54Mn	None.	.....	.....	.....	1.60 Mn	.....	.....
As	0.039	5.009	.....	None.	None.	.....	.....	.....	.....	.....	.....
As_2O_3	.....	.....	0.071	.....	.....	.....	.....	.....	.....	.....	.....
Sb	Trace.	2.697	.....	Trace.	0.12	.....	.....	1.79	.....	.....	.....
Sb_2O_3	.....	.....	0.056	.....	.....	.....	.....	.....	.....	.....	.....
Sn	.....	5.593	.....	.....	.....	.....	.....	.....	.....	.....	.....
SO_3	0.965	.....	0.287	.....	.....	.....	.....	.....	.....	3.00	.....
P_2O_5	2.166	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Cl, Br	.....	.....	0.035	.....	.....	.....	.....	.....	.....	.....	.....
CO_2	Trace.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
CaO	5.361	.....	0.200	6.20	0.30	.....	.....	.....	3.30	.....	.....
MgO	4.297	.....	0.159	.....	.....	.....	.....	.....	.....	.....	.....
Al_2O_3	1.672	0.600	0.328	.....	0.57	18	.....	.....	.....	.....	.....
SiO_2	10.100	2.880	1.577	8.13		.....	.....	.....	17.26	0.40	.....
O	.....	6.256	13.751	.....	.....	.....	.....	.....	.....	.....	.....

(a) Emmons, "Geology and Mining Industry of Leadville," p. 727. (b) Dewey, Bulletin No. 42, "United States National Museum," p. 54. (c) Dobers and Dziegiecki, *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xxxii., p. 102. (d) Mezger, *Berg- und Hüttenmännische Zeitung*, 1853, p. 52. (e) Iles, *School of Mines Quarterly*, xviii., p. 18. (f) Livingstone, private communication, August, 1896. (g) Average for seventeen years Iles. (h) Ounces per ton,

These fumes condense in the upper cooler parts. During their ascent they are partly oxidized, and the oxides may act chemically on the unaltered parts. Thus in wall accretions may be found volatilized metal, metallic sulphides, arsenides, antimonides with their oxides, and secondary products; sulphides being more apt to be formed near the throat of the furnace and oxides near

* Iles, *School of Mines Quarterly*, xviii., p. 18.

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the top of the jackets. Any insoluble silicates found in the accretions come from parts of the furnace lining or from fine particles of the charge.

The finer these particles, the more rapidly do accretions form, thus with galena concentrates, accretions form more rapidly than with lump galena. While they generally form at the sides rather than at the ends of a furnace, it has been noticed in some instances that with the increase of the distance between the tuyeres they form more at the ends and are thicker at the front than at the back.

The Clausthal analysis, where galena is smelted raw, represents a crystallized wall accretion, consisting principally of

	Leadville. (*a*)	Pribram. (*b*)	Pribram. (*b*)
Fe	72.828	48.685	38.875
Pb	18.798	7.309	20.140
Ag	0.1149	0.060	0.160
Au	0.00008	.....	.....
Cu	Trace	0.199	1.107
As	5.083	8.446	2.389
Sb	Trace	0.450	0.735
Mo	0.161	.....	.....
Ni	0.045	Trace	Trace
Co	Trace	Trace	Trace
Zn	Trace	3.610	5.417
Mn	0.015	.....	.....
S	0.650	13.760	18.466
P	0.109	.....	.....
Graphite	0.750	.....	.....
Combined carbon	0.550	.....	.....
Sl, slag, loss	0.900	.....	.....
CaO	.....	0.166	0.780
SiO_2	.....	15.850	9.300

(*a*) Emmons, *Op. cit.*, p. 723. (*b*) Balling, *Berg- und Hüttenmännische Zeitung*, 1867, p. 419.

galena. At Tarnowitz, where slag-roasted galena rich in blende is smelted with the gray slag from the reverberatory furnaces (§ 45), two kinds of accretions form, one a sulphide principally black, the other an oxidized compound having a greenish color. The three analyses by Guyard of Leadville accretions (produced when carbonate ores formed, if not the whole, at least the major part of the charge) show a great variety in composition. The lead, for instance, is present as metal, as sulphide, and as oxide; zinc, arsenic, and antimony as sulphides and as oxides.

The method of removing these accretions, while the furnace is running, and their treatment have already been discussed (§ 77).

§ 91. Hearth Accretions or Sows.—In a furnace with an

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Arents syphon tap these unwelcome products form on top of the lead below the tuyeres; in furnaces tapped from the bottom, they form there. They result from a faulty charge or from a lack of fuel, and are generally mixtures of slag, speise, matte, metallic iron, metallic lead, coke, and charcoal. The metallic iron results from the reduction of ferric oxide, some of it being held in solution by melted matte and dropped when this cools. The iron of a sow is generally carbonized and contains silicon and phosphorus.

It does not usually pay to work up a hearth accretion; it is thrown over the dump or buried, being an eyesore. Flechner* suggests several methods of working furnace-sows. The following, used at the nickel works of Schwerte in Westphalia, (Prussia), is of interest. Furnace-sows containing from 75 to 85% iron, 5 to 8% copper, 3 to 6% molybdenum, 2 to 4% nickel-cobalt, and weighing from 500 to 600 lb. apiece, are gradually melted down with coke on the bottom of a blast furnace, the melted parts running out continuously. In this way a large crust is easily reduced in size, and can then be added again to the ore-charge, where it will be taken up by the speise and the matte. Any mechanical means of breaking up a hearth accretion is sure to cost more than will be recovered from resmelting afterward.

§ 92. Furnace Cleanings and Furnace Refuse.—Furnace cleanings and refuse are a mixture of firebrick, metal-bearing compounds, fuel, etc., obtained in cleaning out a blast furnace when blown down. They are assorted; the waste goes to the slag-heap, the valuable part is added to the ore-bed.

§ 93. Fluedust or Chamber-Dust.†—This product is as important as it is unwelcome. It consists of fine particles of the original charge, and of metals and their compounds that have been volatilized in the lower parts of the furnace and not condensed again in it, but only settle out when passing through condensing flues or chambers.

Table I. gives some older analyses of fluedust and Table II. some more recent ones from the Globe Smelting and Refining Co., Denver, Colo., communicated by Dr. M. W. Iles.

The fluedust from blast furnaces has generally a dark color,

* *Oesterreichische Zeitschrift für Berg- und Hütten-Wesen*, 1889, p. 196.
† Iles, *School of Mines Quarterly*, xvii., p. 97.

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which is caused by the admixture of finely divided fuel. It is black when charcoal forms even a small part of the fuel. The

TABLE I.

	Wyandotte, Mich. (*a*)		Pueblo (*b*)	Harz Mts. (*c*)	Ems. (*d*)		Freiberg. (*e*)		Sheffield. (*f*)		Tarnowitz. (*g*)
Pb	.....	.....	.....	34.8	60.48	67.04	35.02	27.90	.....	PbS 2.25	.....
PbO	19.91	23.77	37.65	18.0	.....	.....	.....	.....	44.80	68.35	26.4
Pb_2SiO_4	.....	.....	.....	2.9	.....	.....	.....	.....	.....	.....	.....
Zn	.....	.....	.....	1.0	3.17	4.23	5.28	49.50	.....	.....	.....
ZnO	0.09	Trace	5.32	1.5	.....	.....	.....	.....	4.80	1.80	57.4
Cu	.....	.....	Trace	.....	Trace	Trace	Trace	.....	+Bi 1.52*h*	.....	.....
CdO	.....	.....	.....	.....	.....	.....	1.30	.....	.....	.....	.....
As	.....	.....	.....	3.0	0.24	0.16	28.08	1.60	3.03*h*	.....	.....
Sb	.....	.....	.....		0.42	0.31	.....	.....		.....	.....
Ag	0.286	0.292	0.04	.....	0.008	0.008	.....	.....	.....	.....	.....
Au	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Fe	.....	.....	.....	1.0	.....	.....	.....	.....	.....	.....	.....
Fe_2O_3	14.43	13.54	24.96	4.5	2.12	1.00	1.57	.....	Trace	.....	.....
Al_2O_3			1.31	.....			.....	.....	10.00	5.40	.....
NiO	0.08	Trace	.....	.....	.....	.....	.....	.....	.....	.....	.....
CoO	0.09	Trace	.....	.....	.....	.....	.....	.....	.....	.....	.....
CaO	8.74	6.62	5.26	.....	1.15	0.61	1.01	7.00	.....	2.68	.....
MgO	3.66	.....	None	.....	.....	.....	0.25	.....	.....	.....	.....
SiO_2	16.11	13.06	8.63	12.3	.....	.....	6.19	.....	9.00	2.25	4.6*i*
S	.....	.....	2.53	7.8	6.22	5.42	.....	.....	.....	.....	.....
SO_3	9.30	8.85	1.61	2.8	14.78	14.07	3.38	18.00	28.81	16.84	4.8
H_2O	1.28	3.76									6.9
CO_2			11.20	.....	.....	.....	.....	.....	.....	.....	.....
C	19.23	22.14		2.5	8.00	5.80	1.17	.....	.....	.....	.....

(*a*) Curtis, "Transactions of American Institute of Mining Engineers," ii., p. 95. (*b*) Dewey, Bulletin No. 42, "United States National Museum, p. 53. (*c*) Balling, *Metallhüttenkunde*, p. 87. (*d*) Freudenberg, *Op. cit.*, p. 19. (*e*) Hering, *Op. cit.*, p. 34. (*f*) French, *Engineering and Mining Journal*, Jan. 17, 1880. (*g*) Kosmann, *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, 1883, xxxi., p. 227. (*h*) As oxides. (*i*) Insoluble residue.

TABLE II.

	Average Sample of All Dust Collected in Ten Years.	Dust from Downcomers of 11 Blast Furnaces.	Dust from Roof of Blast Furnace Building.	Dust from Hood Above Slag Tap.	Fume from Slag-pot While Boiling.	Fume from Reverberatory Settling Furnace.
SiO_2	26.20	3.65	.....	4.80	0.80	.....
Fe	18.40	4.60	.....	.....	.....	.....
Fe_2O_3	.....	.....	2.30	1.40	.....	.....
Mn	0.98	.....	.....	.....	.....	.....
CaO	5.30	2.80	.....	.....	.....	.....
BaO	4.60	.....	.....	.....	.....	.....
Sb	.....	.....	.....	.....	.....	2.20
SbO_3	.....	.....	.....	.....	1.86	.....
As	.....	.....	.....	.....	.....	2.30
As_2O_3	.....	.....	.....	0.64	4.80	.....
Zn	4.74	2.70	.....	.....	.....	26.70
ZnO	.....	.....	.....	32.50	22.20	.....
Cu	0.95	.....	.....	.....	.....	.....
Pb	25.60	47.50	27.10	.....	.....	31.00
PbO	.....	.....	.....	28.50	41.00	.....
$PbSO_4$	.....	.....	.....	27.10	26.20	.....
S	4.20	.....	.....	.....	.....	6.30
SO_3	.....	1.70	.....	.....	.....	.....
Ag, oz.	33.40	16.70	26.40	.....	.....	4.60
Au, oz.	0.14	0.06	0.10	.....	.....	Trace

amount of fluedust formed depends upon a variety of causes. Fine ores or fluxes are carried away easily by the current of

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gases; charcoal, being friable, makes dust; soft coke is broken up to some extent, and causes mechanical losses; and the manipulation of the furnace, affecting the descent of the charge, has a very great influence. Thus careful feeding and cutting out of wall accretions will reduce the forming of fluedust to a great extent. Then, a high temperature in the smelting zone causes much volatilization; in the same way a high blast will cause much vapor to be carried out of the furnace, if the quick ascent of the gases be not checked by the form of the furnace (by having boshes). Extreme figures of the amount of fluedust formed are 0.8 and 15% of the weight of the ore charged.* An average figure is 5%.

The character and consequent value of fluedust changes greatly according to the part of the flue in which it is collected. Near the blast furnace it will resemble very much the ore that was charged, further on it will grow richer in lead and lower in silver until all ore particles have been settled out and only fume remains suspended in the gas current. This will be very rich in lead and very poor in silver. Thus filtered fume coming from blast furnaces producing base bullion assaying from 300 to 500 oz. silver per ton will assay only about 4 oz. This was shown very clearly in the F. L. Bartlett process (§ 53). It may be further illustrated by two instances, one of ample and one of insufficient condensation. The dust collected at a distance of 625 ft. from the blast furnace contained from 20 to 25% lead and 50 oz. silver per ton; after passing the rest of the way, a flue 5,010 ft. long, it contained 65% lead and 1 oz. silver per ton. In the second instance, the dust behind the blast furnace assayed 35% lead and 31 oz. silver per ton; in the main dust-chamber, 41% lead and 26 oz. silver per ton; at the foot of the stack, 52% lead and 17 oz. silver per ton.

The different methods of condensing fluedust‡ may be classed as wet and dry. With the former the gases are either drawn through water alone,‡ or they are pressed through one or more horizontal filters,§ through which fine sprays of water trickle, carrying with them the condensed vapors and filtered solid particles. Steam has been introduced into the gas current with few

* Hahn, "Mineral Resources of the United States," 1882, p. 344.
† Hering, "Die Verdichtung des Hüttenrauches," Stuttgart, 1888.
‡ Eilers, "Transactions of American Institute of Mining Engineers," iii., p. 310.
§ Percy, "Metallurgy of Lead," p. 442.

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good and many bad results, as it helped to convert the sulphur dioxide of the furnace gases into sulphurous and sulphuric acids, both of which readily attack metal, brick, and mortar. Projecting fine sprays of water into and across the current of gases is more effective in condensing fumes, but just as harmful by corroding building materials. Water condensation, while still found in England,* is not much used elsewhere, as it is on the whole unsatisfactory, and the apparatus and its maintenance costly. The wet condensation plant of the St. Louis Smelting and Refining Co., Cheltenham, Mo.,† was successful. It was erected because it became absolutely necessary to condense all fumes and harmful gases. It consisted of a chamber 10 by 20 ft. and 12 ft. high which had two vertical partition walls dividing it into three compartments. The two larger ones were filled with 4-in. drain tiles, placed one upon the other, and received a spray of water from the roof. The gases were made to ascend in the smallest one after having deposited their dust in a large flue, the necessary draft being furnished by a 90-in. Sturtevant fan. On arriving near the top of the chamber, they descended first through one of the large chambers, then ascended again in the other and passed through a long wooden box which was divided by a number of vertical partitions extending from the roof to near the water, which covered the floor. In the middle of the chambers thus formed were water sprays. The gases being forced to travel close to the surface of the cooling water, nearly all the fumes were condensed.

Dry condensation is the almost universal method. It consists in cooling the gases and retarding the velocity of the current. In several instances it has been successfully replaced by dry filtering, which will no doubt in time supersede other methods. Electrical condensation has been experimented with.

The cooling of gases is essential to an effective condensation, as the single particles, being brought closer together, unite more easily into flaky masses, which then settle out. While the gases from a lead blast furnace require very little cooling, the vapors and dust coming from reverberatory furnaces used in smelting and slag-roasting, and in desilverizing base bullion, are apt to be lost if their temperature is not reduced.

* Rösing, *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xxxvi., p. 103.
† Iles, *Loc. cit.*

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Dust-chambers have until recently been almost exclusively built of brick which, being a poor conductor of heat, does not cool effectively. To correct this, Livingstone erected at the works of the Omaha and Grant Smelting & Refining Co., Denver, Colo., dust-chambers of hollow brick which communicated with the open air at one end and at the other with small chimneys erected at intervals on top of the chamber; fresh air being drawn continuously through the terra-cotta walls keeps them cool and thus cools the gases. The gases from 10 blast furnaces pass first through an ordinary brick flue, 10 by 14 ft. and 625 ft. long, then through six terra-cotta flues of the same section built in zigzag, each 835 ft. long, before they pass off into the stack, 16 ft. in diameter and 352 ft. 7 in. high. While at first these flues did excellent work, after awhile some of the fume and dust began to pass through the joints and obstruct the air-passages so that they became partly clogged up, although they are still in use. Hollow bricks are good, if the joints can be made dust-proof.

If the temperature of the gases leaving the stack is reduced to twice that of the surrounding air, the cooling can be considered satisfactory.

An arrangement whereby the gases are satisfactorily cooled by air alone is shown in the following sketch,* Fig. 290. It represents a horizontal sheet-iron flue, with small sliding doors at intervals of 2 ft., suspended by means of iron rods from wooden trestles. It was used at the Richmond Works, Nevada, was 800 ft. long, and ended in a wooden stack 40 ft. high.

A sheet-iron flue of a different form but filling the same purpose, used at the Ems‡ Smelting Works (Prussia), is shown in Figs. 291 and 292. The flues proper, 1 by 0.75 meters (3 ft. $3\frac{3}{8}$ in. by 2 ft. $5\frac{1}{2}$ in.) and 1 meter square (3 ft. $3\frac{3}{8}$ in.), made of $\frac{1}{16}$-in. iron, have triangular projections, the base of the triangle being $6\frac{1}{2}$ ft. long and the height 3 ft. $3\frac{3}{8}$ in.

Suspended sheet-iron flues with a large number of hopper-shaped bottoms are frequently used to-day. Iles‡ recommends painting the flues with "Silicon-Graphite," made by the Joseph Dixon Crucible Co., of Jersey City, N. J.—the paint to be ground

* Eilers, "Transactions of American Institute of Mining Engineers," iii., p. 309.
† Egleston, *Ibid.*, xi., p. 410, plate ii.
‡ *Loc. cit.*

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in oil, thinned down with boiled linseed oil and sufficient No. 2 Dryer to be added to make it dry readily.

A material which has given great satisfaction in Europe for flues and dust-chambers, and which has been introduced into

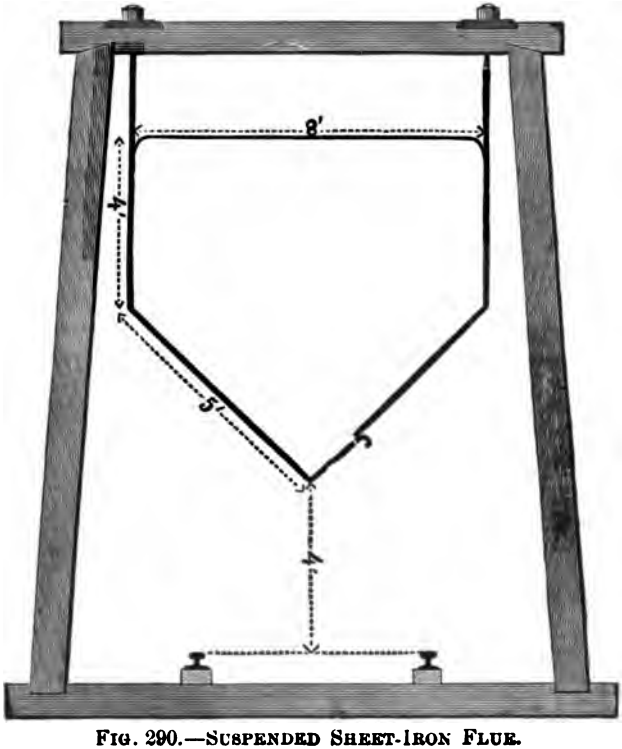
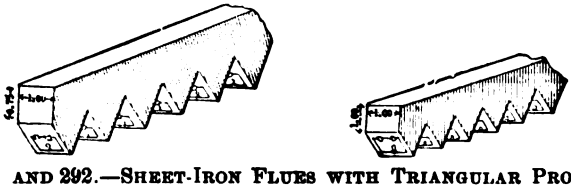


FIG. 290.—SUSPENDED SHEET-IRON FLUE.



FIGS. 291 AND 292.—SHEET-IRON FLUES WITH TRIANGULAR PROJECTIONS.

this country, is that of Monier.* It consists of a coarse lattice of iron wire (Fig. 293) incorporated in cement concrete. The objections at first made to it that the cement would not adhere

* "Freiberger Jahrbuch," 1894. p. 39; *Engineering and Mining Journal*, April 13, 1895; "The Mineral Industry," iv., p. 478.

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to the iron, on account of the different expansion and contraction of the two materials, and that the iron would rust in contact with the wet concrete, have been disproved.

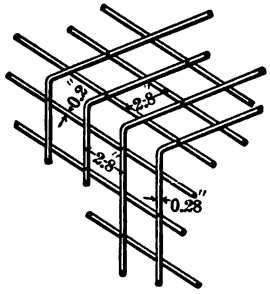


FIG. 293.—LATTICE OF IRON WIRE.

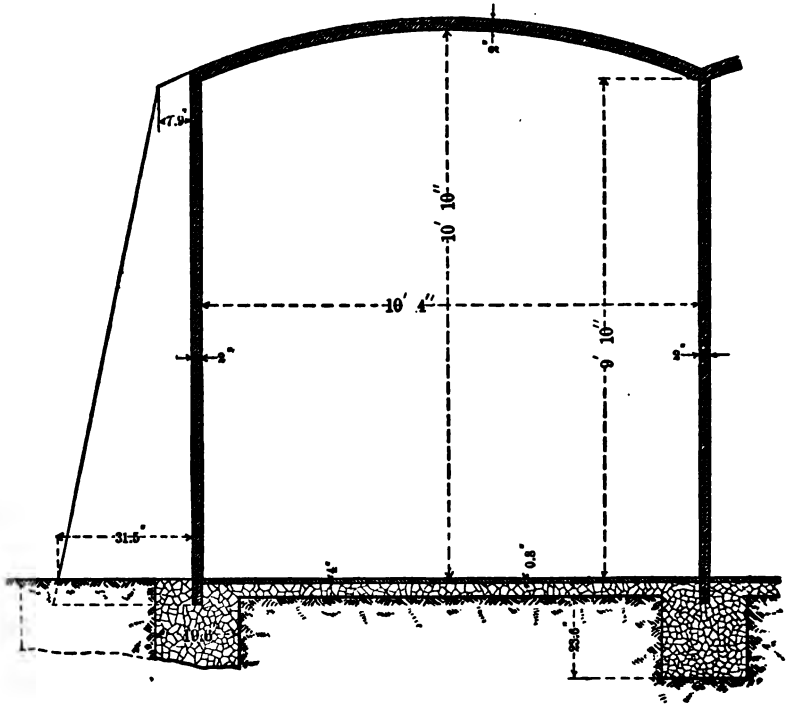


FIG. 294.—SECTION THROUGH MONIER FLUE.

Fig. 294 represents a cross section of a Monier flue erected at Smelting and Refining Works of Freiberg, Saxony. The sides

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rest on and partly extend into a foundation, 1 ft. 11⅝ in. deep and 1 ft. 7¾ in. wide, of cement concrete consisting of 6 parts silicious slag, 3 parts sand, and 1 part cement. The foundation of the floor is only 3½ in. thick. In building the foundation, a V-shaped trough is kept open in the concrete to receive the lower part of the side wall by inserting a board 7⅛ in. wide and 4 in. thick at the top. The flue is put up in lengths of 78 ft. 9 in. The wire lattice (2¾-in. meshes, $\frac{3}{16}$-in. horizontal and ¼-in. vertical wires, with ⅜-in. vertical wires at short intervals) is stretched over the center and kept 1 in. away from it by small supports. The concrete—consisting of 1 part cement with either equal parts of coarse (0.4 to 0.6 in.) sand and a mixture of coarse sand and fine (30-mesh and smaller), or 1 part of the former and 2 of the latter—is pressed through the wire lattice, and when the space under it has been filled, it is tamped on 1 in. thick, making the entire thickness of the wall 2 in. In a few days it is hard enough to permit the removal of the center. In order to gave stability to the flue, buttresses of Monier material are placed at intervals of 6 ft. 3¾ in. After finishing the walls, the bottom is tamped down with concrete to a thickness of 3½ in. The outer and inner sides of the flue are then coated with a thin cement mortar, the bottom receiving a coating $\frac{2}{3}\frac{1}{2}$ in. in thickness. Finally the whole interior is painted with an acid-proof paint, the composition of which is not made public, and the exterior with tar. One sq. meter (10.7641 sq. ft.) of this Monier wall with supports 1 meter (3.28 ft.) apart can bear a load of 720 kg. (1,587 lb.) if evenly distributed and half the amount if applied in the center.

The flues have stood well the corroding influences of the gases. The temperature of the gases after passing through the zigzag flue, which has a total length of 682 ft. 3 in., is reduced 21° C.

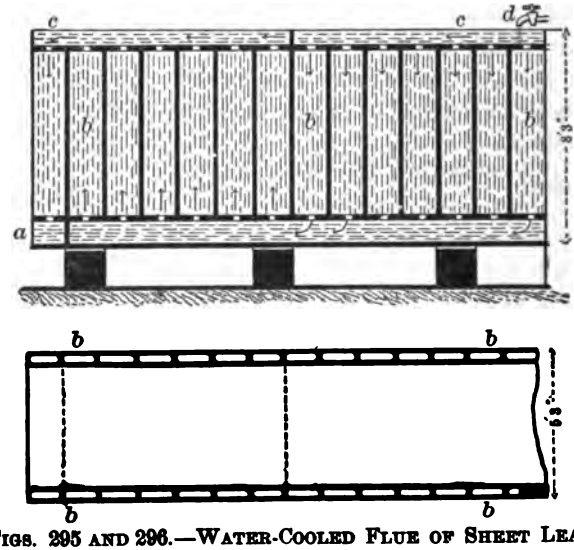
The following data, based on results obtained at Freiberg, show the cooling effect of the different materials used in building flues. The temperature of gases is reduced 1° C. by 9 ft. 10 in. of lead flue, by 14 ft. 9⅛ in. of Monier flue with two sides exposed to the air, by 19 ft. 8¼ in. of brick flue with two sides exposed, by 27 ft. 2¾ in. of Monier flue with one side, and by 39 ft. 4⅜ in. of brick flue with one side exposed.

To make the cooling more effective and thus shorten the flue, water has been used in different ways. In 1877 Hagen* intro-

* "Freiberger Jahrbuch," 1879, p. 151; Hering, *Op. cit.*, p. 20, and private notes, 1890.

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duced in Freiberg his water-cooled flues made of sheet-lead (Figs. 295 and 296). The roof (5 ft. 3 in.) and sides (8 ft. 2¾ in.) are suspended from a wooden framework in the same way as with a sulphuric-acid chamber. The top *c* is cooled by a slight flow of water *d*. This circulates along the sides through elliptical pipes *b*, soldered to the flue and to each other. Fig. 295, a vertical longitudinal section through the pipes *b*, shows the circulation of the water. This arrangement permits cooling with less water than would be required if it overflowed from the roof down the sides.



FIGS. 295 AND 296.—WATER-COOLED FLUE OF SHEET LEAD.

Another device for cooling gases is the "Water Tower" in use at Tarnowitz, Silesia. It is an oblong brick tower, 16 ft. 6 in. by 6 ft. 6 in. in the clear and 32 ft. 10 in. high, closed at the top with a perforated cast-iron plate strengthened by ribs. From each of the 180 holes in the plate is suspended a bundle of cooling pipes,* shown in Fig. 297, consisting of a central water-inlet pipe *c* 16 ft. 6 in. long and 1⅝ in. in diameter, surrounded by 6 smaller pipes ⅝ in. in diameter, through which the cooling water ascends and passes off. The lower ends of the pipes of

* Saeger, *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, 1893, xlv., p. 280.

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a bundle are screwed into a cast-iron cap with cleaning-hole *g*, the upper ends pass through a similar cap *k*, the collar of which rests on the cover *e* of the tower, a rubber gasket forming a water-tight joint. Any dust adhering to the pipes is removed periodically by a jet of steam introduced through a movable central pipe having nipples at certain intervals. The gases from six blast furnaces, 51½ in. in diameter at the tuyeres, enter the tower near the top at one of the sides and pass downward; they then zigzag through a Monier flue having vertical partitions of corrugated iron and 119,000 Roesing wires, 10 ft. long and of

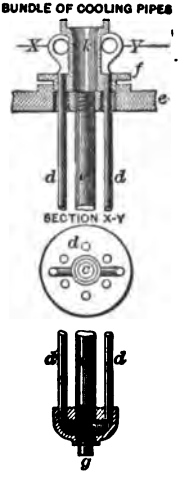


FIG. 297.—BUNDLE OF COOLING PIPES OF THE WATER-TOWER AT TARNOWITZ, SILESIA.

No. 10 wire gauge, suspended in it (see below) and travel through an underground flue to a suction fan which finally forces them through two Richter towers of sheet-lead, 23 ft. high and 10 ft. by 6 ft. 6 in., into the stack. The temperature of the gases when entering these towers is 200° C., when leaving 50° C.

Another arrangement was introduced at Freiberg by Richter.* It consists in letting the gases ascend in a set (five) of wooden towers, lined with ⅛-in. sheet-lead. These are 18½ ft. high and 6½ by 10 ft. in the clear. Water trickles down in the towers, each of which contains ten rows of small roofs, supported by

* "Freiberger Jahrbuch," 1889, p. 57; *Berg- und Hüttenmännische Zeitung*, 1890, p. 129; *Engineering and Mining Journal*, Feb. 15, 1890, and private notes, 1890.

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lead-coated rails. Each roof consists of three sheets of hard lead, 3 ft. 3½ in. long and $\frac{5}{32}$ in. thick, bent to the form of a V, and having incisions 1⅛ in. deep at the lower border, in order that the water may run off freely and leave the necessary interstices for the ascending gases. The roofs are placed so that the discharge of one set strikes on the ridgepoles of the one next below. The water for the five towers is drawn from two large tanks, and a short distance above the lowest row of roofs is a coarse grating, which is to distribute the water evenly in order that the gases, on striking the first roof, may be cooled as much as possible. The temperature of the gases on entering the towers is from 100 to 115° C., and on leaving from 40 to 60° C.; they then pass through a leaden flue 1,673 ft. in length before entering the main stack, which is 453 ft. high. Each tower can be separately shut out from the main flue, which is done at certain intervals, to remove, by means of a stream of water, the fluedust that collects beneath the roofs.

The main effect of this apparatus is due not so much to the larger surface it offers to the gas current as to the fact that it cools the gases. Five times as much fluedust is collected with it as before. This system given up at Freiberg has been, as shown above, added to the condensation plant of Tarnowitz. It is in use in this country at a pyritic smelter to condense the small amount of silver-bearing lead-fume which would otherwise be lost.

Other arrangements for cooling gases are those by Schlösser and Ernst,* who let cooled solutions circulate in coils of iron pipe placed at certain intervals in the main gas-flue.

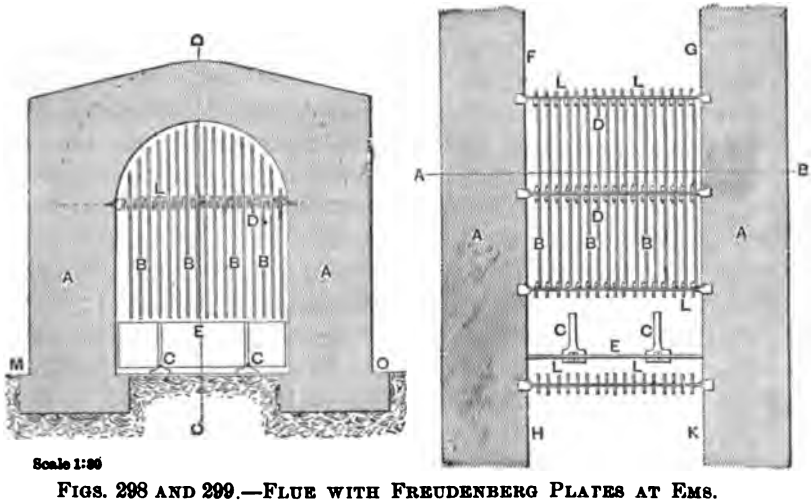
In settling out fluedust the most important thing after cooling is the retarding of the velocity of the air-current and the exposing of a large condensation surface to it. It has been found that the point of least friction is on the center line of a flue at 0.7 of its height. Here the air has its greatest velocity. This diminishes very little upward, but so much downward that there may even be a small counter-current at the bottom of a flue. The greater part of the dust collects on the floor, but some adheres to the roof. The speed of the air-current can be retarded by increasing the volume of the flue, by changing its direction, or by increasing its surface.

* *Berg- und Hüttenmännische Zeitung*, 1885, p. 464; 1887, p. 434.

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Increasing the size of flues through which the gases pass has always been considered a very effective means of settling out fluedust. On entering an enlarged chamber the velocity of the current is gradually slackened to the point where the draft near the exit begins to show its effect; then there is a gradual increase of velocity. On this account a comparatively small part of the enlarged flue or chamber is really useful in the settling out of the dust.

The reason that a change in the direction of the current is also not so effective as might be expected is that, though at the turn-



FIGS. 298 AND 299.—FLUE WITH FREUDENBERG PLATES AT EMS.

ing-point the speed of the current may be slackened for a short distance, the pressure of the air behind and the draft in front will quickly restore it to the normal rate.

It is Freudenberg's* great merit to have discovered that an increase of surface is the most effective means of settling particles held in suspension by a current of air, and that the amount of fluedust settled out stands in direct proportion to the area of surface with which it comes in contact. This increase of surface means increase of friction between the stationary surface and the moving current of gas, a consequent retarding of its velocity,

* Freudenberg, "Die auf der Bleihütte bei Ems zur Gewinnung des Flugstaubes getroffenen Einrichtungen," Ems, 1882; Abstract, *Engineering and Mining Journal*, July 1, 1882; Egleston, "Transactions of American Institute of Mining Engineers," xi., p. 379; Stetefeldt, "Comment on Freudenberg's Plates," *Engineering and Mining Journal*, July 28, 1883.

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and with it a settling out of dust particles. The surface also attracts these, if it is cold.

Aitken* has shown that a hot surface repels them, especially if moist. This explains how the introduction of vaporized or finely divided water† into the air-current has not proved so effective as was anticipated.

In order to increase the surface, Freudenberg suspended thin sheet-iron plates parallel with the air-current; and to prevent the dust from being carried off, after it had once settled out, he placed partitions across the bottom, reaching nearly to the hanging plates. In Figs. 298 and 299, representing a vertical and a horizontal section of a flue, are seen the sheet-iron plates *B*, of $\frac{1}{32}$-in. iron and from $4\frac{3}{8}$ to $5\frac{7}{8}$ in. apart, having pieces of iron *D* bent to the form of a hook, riveted to their ends. By these they are suspended from pins passing through rectangular cross-bars *L*, which, twisted flat at both ends, reach $1\frac{1}{8}$ in. into the side walls *A* of the flue. The plates are self-cleaning, *i.e.*, the fluedust collecting on the plates falls off when it has grown to a thickness of $1\frac{1}{8}$ to $1\frac{1}{4}$ in. and falls to the bottom. Every 16 or 20 ft. cross partitions *E*, $7\frac{3}{4}$ in. high, protect the lower part of the flue from the air-current. They are bolted to cast-iron supports *C*.

At Freiberg‡ the plates are differently supported, as shown in Figs. 300 to 304. All plates are 3 ft. $3\frac{3}{8}$ in. wide and $\frac{1}{32}$ in. thick. The plates in the arched portion of the flue (from 8 to 16 in number) are from 1 ft. 3 in. to 2 ft. $10\frac{1}{4}$ in. long. They are suspended at intervals of $4\frac{3}{4}$ in. by hooks from pins passing through band-iron which, following the sweep of the arch, is let in at the sides and fastened to the roof. There are half as many plates in the lower part of the flue; they are 3 ft. $7\frac{1}{4}$ in. long and about 8 in. apart. They rest with lugs, riveted to the upper ends on cross-bars let into the sides. Thus when enough dust has collected on the bottom of the flue to make a clean-up, the lower plates are pushed to one side and then do not interfere with the work.

A comparison of three years' work both with and without Freudenberg plates shows that the amount of fluedust collected

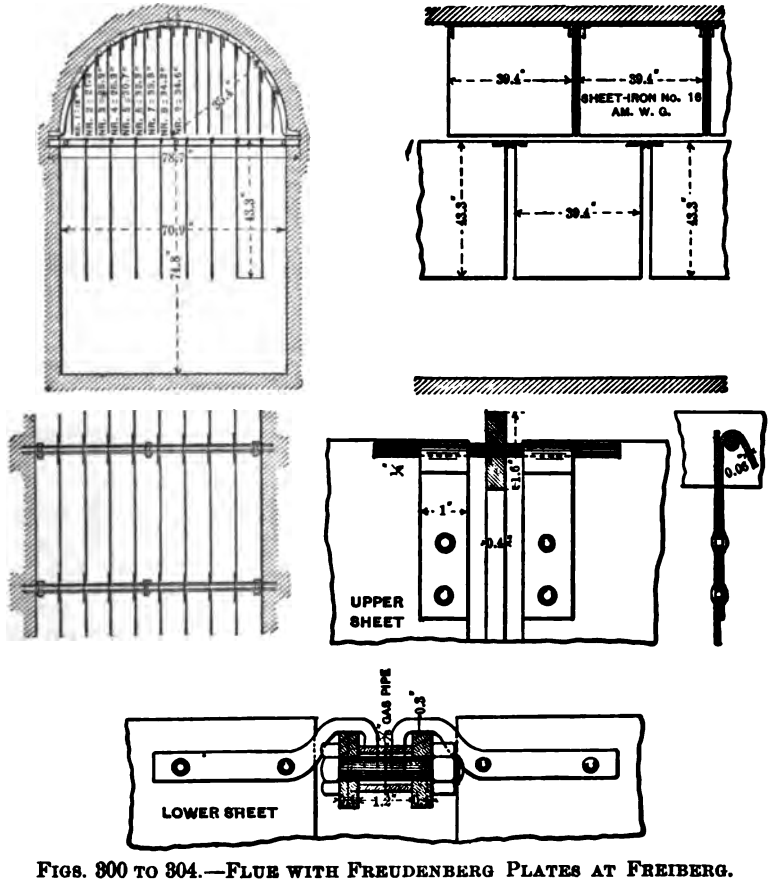
* "Proceedings of the Royal Society of Edinburgh," xxxii., p. 239; *Wagner's Jahresberichte*, 1884, p. 1307.

† Iles and Keiper, *Engineering and Mining Journal*, Feb. 27, 1886.

‡ "Freiberger Jahrbuch," 1894, p. 39; *Engineering and Mining Journal*, April 13, 1895 "The Mineral Industry," iv., p. 476.

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both in the flues leading to the chamber containing the plates and in the chamber itself is twice as much with the plates as without them, and the dust in the flue leading from the chamber to the stack much less. A comparison of the assay values shows that the dust settled out in the flue leading to the chamber was



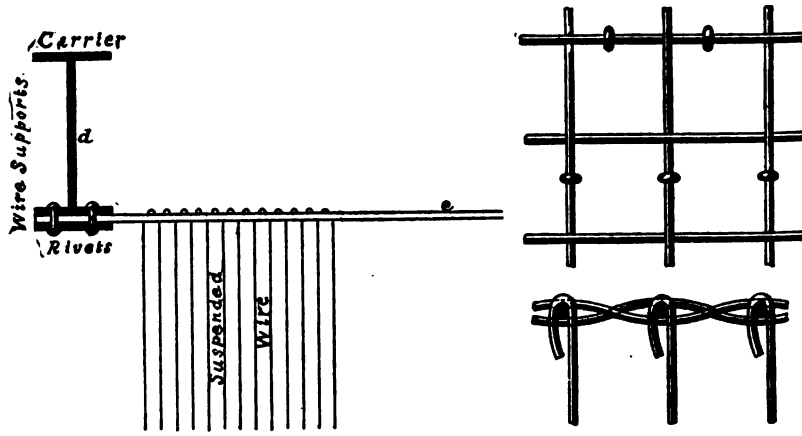
FIGS. 300 TO 304.—FLUE WITH FREUDENBERG PLATES AT FREIBERG.

richer in lead and arsenic than when no plates were used, and that in the chamber proper and the flue leading from it to the chimney more arsenic was collected and less lead. These results speak very well for the plates. The disadvantages are that they are not suited for high temperatures and are quickly corroded at temperatures below 50° C. by the acid moisture which condenses

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on them; that they very much retard the natural draft, so that forced draft becomes necessary; and that they are expensive both as regards first cost and royalty.

The investigations of Freudenberg form the basis for the construction of most modern dust-chambers. Rösing‡ has substituted iron wires for Freudenberg plates. The wires are of No. 10 gauge and 10 ft. long. The manner of suspending them is shown in Figs. 305 to 307. These represent a wire screen *e* of about 1-in. mesh, riveted to I-beams *d*, running along the top of



FIGS. 305 TO 307.—THE WIRE SYSTEM OF ROESING.

a flue which is 16 ft. 6 in. high. From the screen the wires are suspended. By a shaking arrangement the dust can be more easily removed from the wire than from the plates. At Tarnowitz, Silesia, during 300 working days in 1888 there passed off into the open air without the wire system 121,000 lb. of solid matter† for every furnace; with the wire system the weight of the solid matter was reduced to 90,200 lb. without diminishing the draft of the chimney. In Hering's book, already quoted, are given numerous sketches of condensation chambers that have been projected or built since Freudenberg's results became generally known.

Filtering of gases through coarsely woven muslin or woolen cloth, which was formerly confined to the Lewis-Bartlett process

‡ *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, 1893, xli., p. 267.

† The method of determining this is described in the reference.

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and to the manufacture of zinc white either directly from the ore or from the burning of zinc vapor, has been introduced and is working satisfactorily at several works. The earlier attempts at drawing furnace gases through towers filled with brushwood, coke, etc., have all proved failures.

The Lewis-Bartlett filter process, as carried out at Joplin, Mo., has been described and illustrated in § 52. The description of the filtering of fume at Cañon City, Colo., at the works of the American Zinc-Lead Co., (§ 53) gives the leading data necessary for the construction of a bag-house in which the gases are filtered through suspended bags. The only silver-lead works using bags, so far, for filtering are those of the Globe Smelting and Refining Co., Denver, Colo., but no details can be given. The bags had been tried many years ago at the Grant Smelter* at Leadville, Colo., but were given up again, as the dust saved, while running high in lead, contained only from 6 to 10 oz. silver per ton.

At the Omaha works† of the Omaha and Grant Smelting and Refining Co., the Brown-De Camp fume collector is doing satisfactory work. The fluedust is collected by filtering through cloth stretched up and down zigzag across a condensing chamber, the cloth offering a very large filtering surface to the dust-laden gases which are blown against its lower sides by a fan. In order to keep the pores open, the dust collecting on the lower sides is shaken off at intervals by beaters. The gases from 10 blast furnaces, 42 by 120 in. at the tuyeres, are sucked by a 12-ft. Murphy fan making 200 revolutions per minute through a flue 800 ft. long and 64 sq. ft. in area, where they cool down to 65° C. and then pass into the collector. From seven blast furnaces, 42 by 120 in., are recovered with a filter-cloth area of 80,000 sq. ft. 100 tons lead per month, assaying 10 oz. silver per ton, amounting to about 6% of the lead of the charge. The finely divided dust, containing considerable carbon, is transferred from the condenser into a closed chamber and piled up and ignited, where the carbon burning off slowly, as in the Lewis-Bartlett process, will create sufficient heat to agglomerate the dust and thus make it suited for charging back into the blast furnace. No further details can be had.

* Guyard in Emmons' "Geology and Mining Industry of Leadville," monograph xii., U. S. Geological Survey, pp. 673, 717.

† *Engineering and Mining Journal*, Oct. 19, 1895; "The Mineral Industry," iv., p. 476.

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The Williams screen-filtering apparatus* consists of a sheet-iron cylinder mounted on columns. It is closed at the bottom by a conical hopper with discharging door, and has at the top an outlet for the gases. At the side, below a horizontal wire screen, is the air inlet. In the cylinder is suspended a sheet-iron funnel with a long neck. The mouth is above the wire netting partition; the neck is below and reaches to near the discharge opening of the hopper-shaped bottom of the outer cylinder. The area of the inlet pipe is equal to the annular space surrounding the top of the funnel. Dust and fume-laden gases enter below the wire partition, pass up through it, and are deflected downward before they ascend through the pipe in the roof. The coarse dust, separated out by the screen, and the finer dust dropped by the change of direction of current, are collected in the hopper of the sheet-iron cylinder and discharged at intervals.

It has been found that the apparatus worked satisfactorily without the screen, so this has been omitted.

While the different filtering devices work very well on gases from blast furnaces by saving all the ore dust and lead fume, they unfortunately prevent other vapors, *e.g.*, those of arsenic, from passing off into the open air. This is their great disadvantage. The filtering of gases from roasting furnaces has not yet been made commercially successful, as the sulphuric acid they contain quickly corrodes all known kinds of filtering cloths.

Many attempts have been made to reduce the sulphuric acid to less harmful sulphurous acid, but, at least as far as the writer is aware, without success. It would seem, therefore, that wet condensation while out of place for fumes from lead blast furnaces was quite desirable in connection with roasting furnaces, the fumes of which are at present going to waste.

The use of electricity† in settling fluedust has not been so successful as the first experiments seemed to promise. In closed chambers static electricity quickly clears dust-laden air; when, however, the air is in motion electricity produces no effect.

As regards the arrangement of the plant, it is important to have a dust-chamber near the furnace, as most of the heavy dust is set-

* United States Patent, No. 554,563, Feb. 11, 1896; "The Mineral Industry," v., p. 410.

† Hutchings, *Berg- und Hüttenmännische Zeitung*, 1885, p. 253; *Engineering and Mining Journal*, May 8, 1886; Rösing, *Berg- und Hüttenmännische Zeitung*, 1885, p. 290; Bartlett, *Engineering and Mining Journal*, March 13, 1886.

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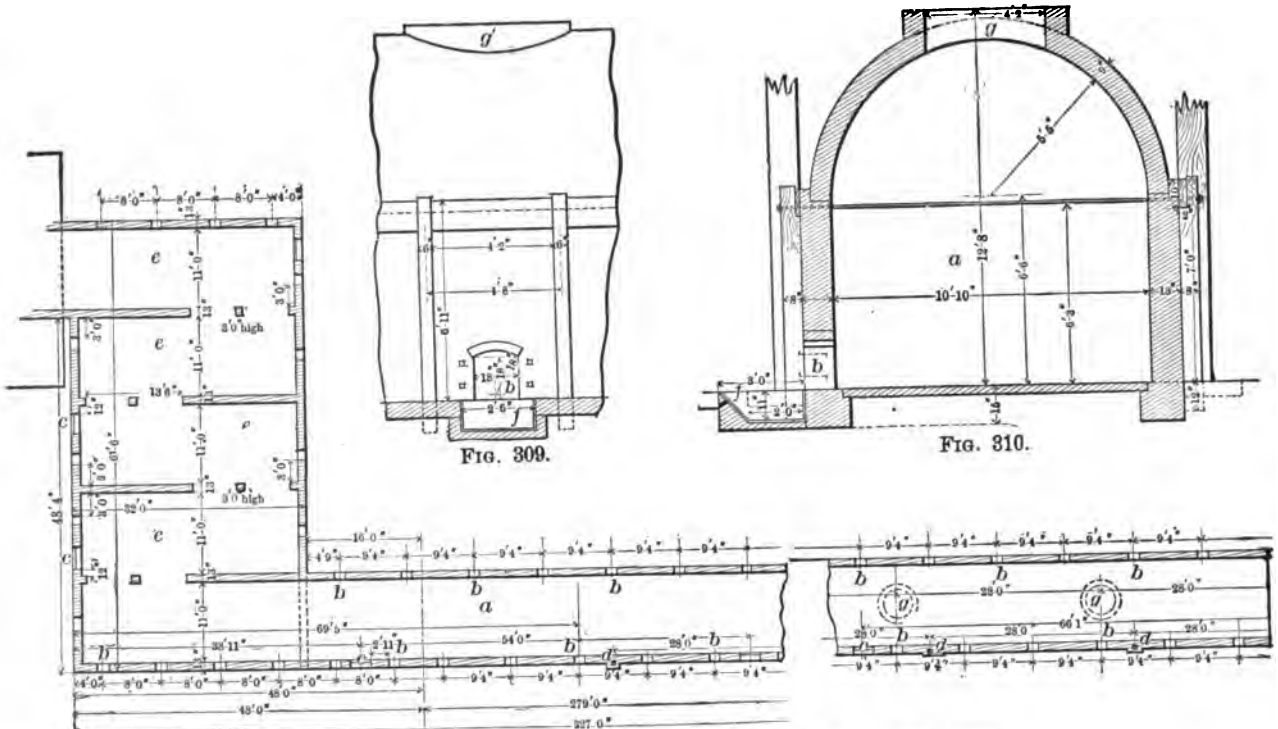


FIG. 309.

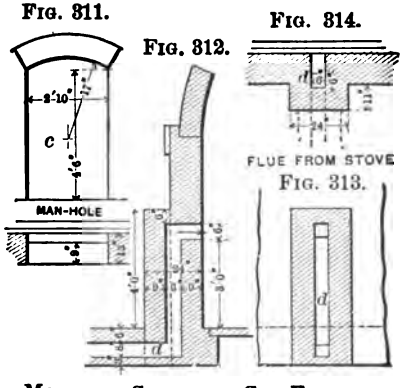
FIG. 310.

FIGS. 308 TO 310.—MONTANA SMELTING CO., FLUE AND DUST CHAMBER.

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tled out quickly. If the chamber has to be some distance away, as is often the case, the conduits will be sheet-iron pipes with hoppers along the lower side. If the dust and fume are not to be filtered, the chamber will lead into a flue (or flues) with Freudenberg plates, where the gases will drop most of their dust and then pass off through a chimney. Sometimes it will be necessary to have a fireplace at the foot of the chimney to create the required draft. Again, a fan in the flue may be needed to suck and push onward the gases.

If the dust and fume are to be filtered, the gases will be drawn by a suction fan from the chamber through a long cooling flue and forced through the filter cloth. In order to assist the cool-



FIGS. 311 TO 314.—MONTANA SMELTING CO., FLUE AND DUST CHAMBER.

ing and to dilute the gases cold air will be sucked into the cooling flue at one or more points, thus avoiding danger from fire in case of explosions, etc.

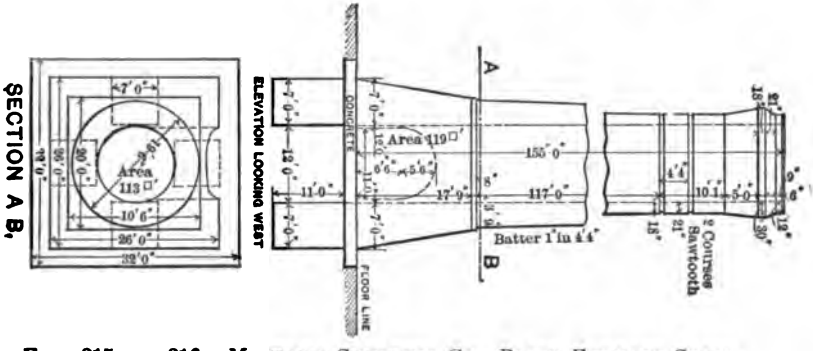
The dust-chamber of the Montana Smelting Co.s' Works at Great Falls, Mont., shown in Figs. 308 to 314, may serve as an example of the manner in which the dust is now very frequently collected at lead smelters. A general plan and elevation of the works has been given in Figs. 152 and 153. In Fig. 308 is shown a horizontal section of the main flue *a*, which runs along the back of the furnace floor. It receives the dust-laden gases from the blast-furnaces through openings *g* in the roof. The gases pass from the flue to the chambers *e*, and then into the stack.

If Freudenberg plates were used, they would be suspended in

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that part of the flue reaching from the chamber *e* to the first blast furnace *g*.

The main flue has at intervals of 9 ft. 4 in. small openings *b* (Figs. 308, 309 and 310), placed diagonally opposite each other, through which the fluedust is raked out periodically into the shallow pits *f*, where it is moistened before being handled. They are closed with hinged iron doors. Every 56 ft. (the distance between the centers of two blast furnaces) there is a manhole *c* (Figs. 308 and 311), which is bricked up with a half course of brick. These are easily removed when the flue is to be entered. The floor of the flue is level. Frequently they are inclined toward the drawing doors, the inclination being often so great that most of the dust will slip down of itself into a pit (similar to *f* in Fig. 310),



FIGS. 315 AND 316.—MONTANA SMELTING CO., BLAST FURNACE STACK.

when the door is raised, and require very little raking. For each blast furnace will be found in the side wall a small opening *d* (Figs. 308, 312–314). This is connected by an underground flue with the stove that heats the cast-iron lead-pot placed next to the lead-well (§ 75). From the flue *a* the gases pass four dust-chambers *e*, having vaulted arches running parallel to that of flue *a*. The area of the passage from the main flue to the first chamber and of the one between the single chambers is somewhat contracted. The gases must thus pass the openings in the partition walls with an increased velocity, which diminishes again as soon as they enter the next chamber. The last chamber ends in the stack, of which Figs. 315 and 316 give the details.

Dust-flues and chambers are usually built of brick, and require strong walls and close binding to keep them air-tight.

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§ 94. TREATMENT OF FLUEDUST.*—The question how to treat fluedust with a minimum of loss is a difficult one. Many suggestions have been made and various methods tried.

To wet down fluedust and put it back into the furnace is of no use, unless it contains some soluble sulphate, as when dry it will simply be blown out again. The only sulphate of any importance in fluedust is zinc sulphate. Fluedust containing from 1 to 2% zinc will, when moistened, harden easily. This is especially the case with the light-colored dust from roasting furnaces. The finely divided carbon in the blast furnace dust hinders to some extent this hardening. So by mixing the two, satisfactory bricks can be obtained. An analysis of such a brick gave Livingstone: SiO_2 17.1%, Fe 9.6%, S 8.7%, Zn 3.0%, Pb 28.4%, CaO 2.8%. Another sample of bricked dust gave Dorr 3.04% zinc oxide. Treatment with water gave SO_3 0.35%, Zn 0.04%; with ammonium sesquicarbonate SO_3 1.70%, ZnO 0.04%; with acetic acid SO_3 trace, Zn 0.04%.

The binding property of zinc sulphate was observed by the writer many years ago while roasting blende-bearing galena to obtain zinc sulphate for leaching. After removing as much of the zinc as was possible, the residual ore, when dry, became so hard that it required strong blows with a sledge to break it. But the ore retained more than 1 or 2% Zn.

Hahn† found that fluedust mixed with a solution of ferrous sulphate and formed into bricks (by hand) became so hard when sun-dried that it would stand much handling. Perhaps an addition of slacked lime might prove useful, as it precipitates the ferrous oxide, which, on being exposed to the air, becomes peroxidized and forms hard lumps.

A favorite method of compacting fluedust is to mix it with lime and form it into bricks which when hardened are added to the furnace charge. Formerly from 8 to 10% of slacked lime was used and the bricks were made by hand. Cahen‡ gives an example of mixing fluedust (with 44% lead) and slacked lime. The bricks formed were smelted alone in the blast furnace with 74% of slag, and only 9.60% of the lead charged was lost.

Considerable advances have been made within the last few

* Iles, *Engineering and Mining Journal*, Jan. 30; Feb. 6, 27, 1886.
† "Mineral Resources of the United States," 1882, p. 344.
‡ "Métallurgie du plomb," Liége, 1863, p. 102.

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years in the bricking of fluedust; they consist mainly in not bricking it alone, in the use of machinery, and in burning the bricks.

It has already been stated that blast-furnace dust containing zinc sulphate will not harden as readily as that from roasting furnaces on account of its carbonaceous matter, and for the same reason a binder, such as lime, added to it will not act well. Hence more lime will be required for blast-furnace dust than for roaster dust. Going a step further, the binding power of lime for roaster dust will be about the same as that for raw ore of the same degree of fineness and content of sulphur, as of course pyritic ores rich in sulphur, *e.g.*, concentrates, cannot be bricked, because upon heating they will fly to pieces. If, then, ore crushed to pass a limiting sieve of say 2 meshes to the linear inch be mixed in with fine ore and fluedust, the additional advantage is gained of the angular shape of the crushed ore, which will require less lime for binding and will dry more satisfactorily on account of the mass of capillary spaces between the crushed dry ore, the moist fluedust, and the cream-like binder. When the brick is being dried, either in the open air alone or additionally in a heated chamber, it will, on account of its porosity, give up readily its water and shrink without cracking. When charged into the blast furnace part of it will again fall to pieces, but to a much smaller extent than if the fluedust had been bricked alone with a high percentage of lime. If the dried brick, before charging into the blast furnace, is burned in a kiln, it will, when the temperature has been raised to the point where carbon and sulphides begin to roast, burn readily and the heat generated will cause the lead or matte to cement together to a solid mass the single particles of a brick. This gives the explanation for the present practice of bricking fine silicious ores with from 2 to 5%, or on the average 3% of slacked lime and 20% or more fluedust to frit together the single particles when exposed to heat. With ores having a clayey gangue, or with roasted ore or roasted matte, no addition of fluedust is necessary, if the bricks be kiln-burned; they will frit together forming solid bricks and often as much as 50% of the sulphur of the roasted material will have been eliminated. With silicious ore free from sulphur, from 2 to 3% fine coke is added to the mixture to furnish the fuel necessary for burning in the kiln.

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The bricks consisting of silicious ore and fluedust weigh about 8 lb. apiece, those from roasted sulphide ores or matte about 12 lb.

In order to brick ores satisfactorily and cheaply with a small percentage of lime, it is necessary to subject them to a high pressure, which requires machinery. Different forms of brick presses are used to-day, which very soon pay for themselves on account

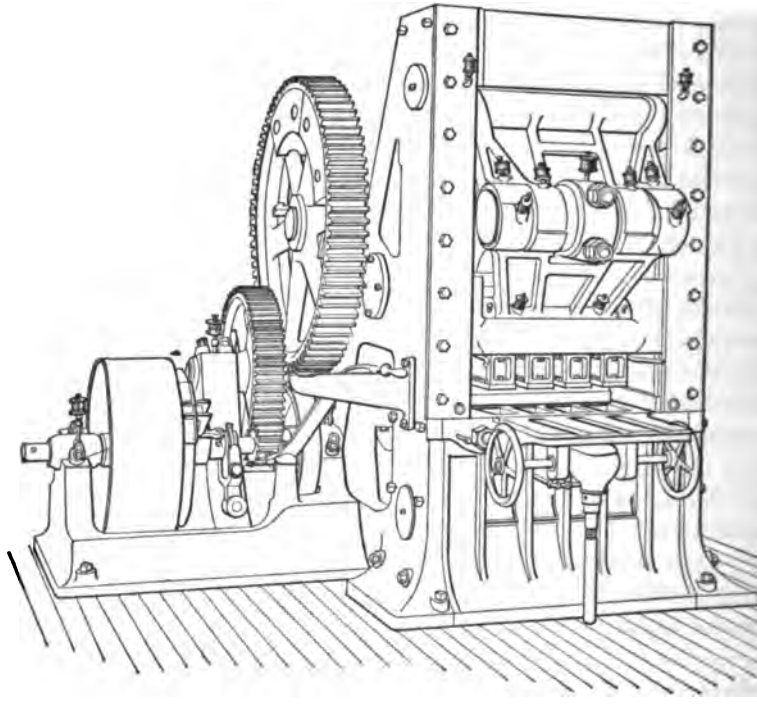


FIG. 317.—THE BOYD BRICK PRESS.

of the excellent work that they do. Two may serve as examples: The Boyd brick press* is a machine used by some of the leading smelters. Fig. 317 shows a four-mold press. The framework of the machine consists of two cast-iron housings bolted together, the mold-table locking into the frame. This carries at the back the bearings of the different shaftings and forms a guide for two

* *Engineering*, 1893; No. 24, and private notes, 1896-7; manufactured by Messrs. Chisholm Boyd & White, Chicago, Ill.

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inner side-bars which take up all the strain due to pressure. The bottom is planed for convenience of setting, no foundation bolts being required. The press proper consists of the side-bars, a pair of toggles and a cross-head. The side-bars are connected at the top by a steel pin journaled in them, at the bottom by bolts passing through the mold-table; the toggles receive their movement at the middle joint from a connecting rod operated by a crank-shaft, and exert a pressure of 600 tons or 150 tons on each of the four bricks; the cross-head with the upper steam-heated plungers is guided by the side-bars; lastly there is the mold-table with the lower steam-heated adjustable plungers which, rising in the mold, lift the finished brick and the device for pushing it to the front table.

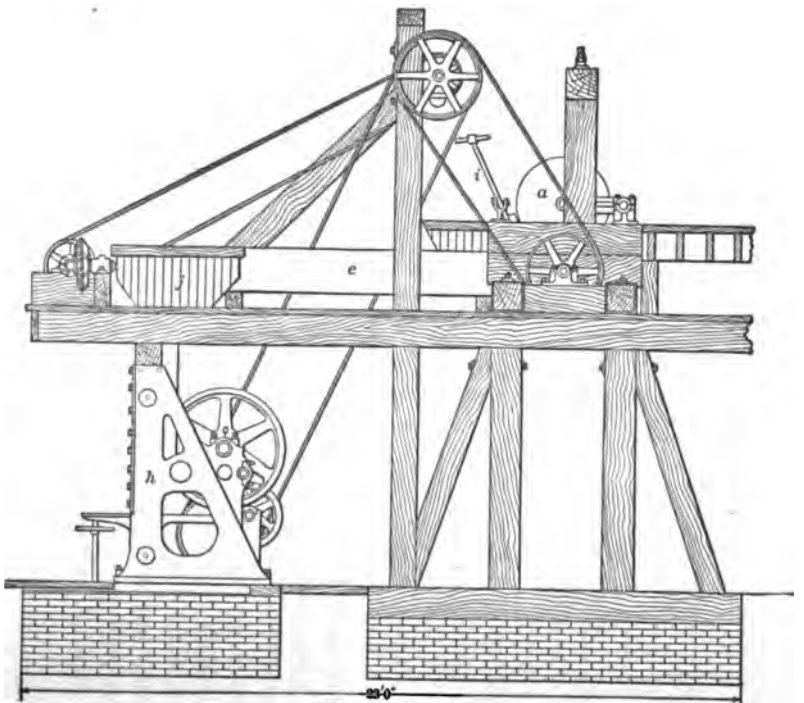
The method of working is as follows: The bricking material mixed with the necessary lime and water is delivered to the molds through canvas spouts, the feed is shut off automatically and the top of the mold smoothed. The mixture is then pressed by the straightening of the toggles and the brick raised to the top of the mold (when the toggles have been again opened) and pushed to the front to be removed by the attendant; the feed now starts again, and the process is repeated. Thus the entire operation from feed to delivery is automatic.

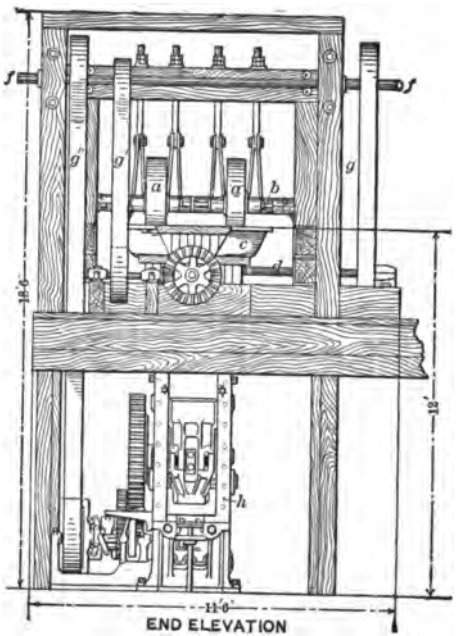
The machine weighs 29,500 lb., occupies a floor-space of 7 by 9 ft. and is 8 ft. high; it has a capacity of from 20,000 to 30,000 bricks of ordinary size in 10 hours and requires a 30-horse-power engine.

Six—three—and two-mold presses are found in the market.

The plant required for bricking is simple. It consists of a number of lime-slacking bins, an edge mill, a horizontal pug-mill or disintegrator, and a press. The lime-slacking bins vary in size and number. For a 6-mold press six bins are sufficient. They are made 10 by 7 ft. and 2 ft. deep, or 4 by 6 ft. and 1½ ft. deep. They are, of course, placed between the beds containing the material to be bricked and the machinery. The machinery of a two-mold bricking press is shown in Figs. 318 and 319. In the edge-mill the rollers *a* revolve on the stationary shaft *b* by the tangential friction of the pan *c*, which is driven by a pinion on shaft *d* and a bevel gear on the bottom of the pan. The pug-mill or disintegrator *e* consists of a horizontal trough in which a shaft bearing knives is rotated by a vertical bevel gear *kk'* in

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FIGS. 318 AND 319.—BRICKING PLANT WITH BOYD PRESS.

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the same manner as in the pan. The power is transmitted from the main shaft *f* by belting *gg'g''* to the pan, the pug-mill and the press *h*, which has already been described. The mode of operating is as follows: From the bedding-bin the ore is charged into the pan, about 3% slacked lime is added and the whole is mixed under the rollers for a few minutes until the mass has assumed a uniform color. The water of the slacked lime is as a rule sufficient to moisten the mixture, which has the consistency of slightly watered brasque. The pan contents is discharged by

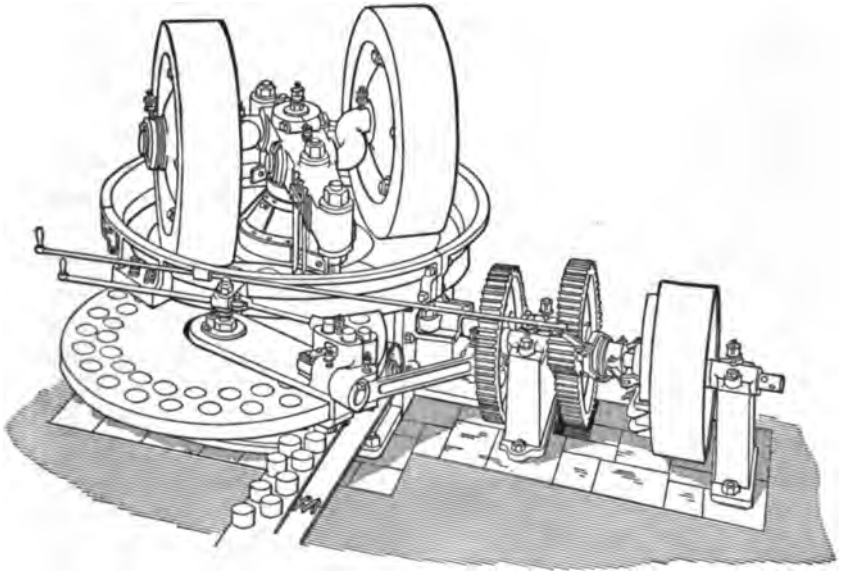


FIG. 320.—THE WHITE MINERAL PRESS.

means of the shovel *i*, mounted on the frame of the pan, into the pug-mill. This breaks up any lumps that may have formed in the pan and furnishes a uniform stream of thoroughly mixed and evenly moistened mixture to the hopper *j* of the brick press.

The labor required to operate a 4 or 6-mold press is two men to bring the mixture from the bed, two at the roller-mill two to remove the brick to the drying-floor (often an open box-car), and one foreman. With a 2-mold press half the number of men will be necessary.

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The White Mineral Press.*—The main parts of this machine, shown in Fig. 320, are a roller mill and a molding disk. The pan of the mill is 7 ft. in diameter and is lined in the path of the rollers with chilled iron plates; the rollers are 48 in. in diameter, have a 10-in. face, are provided with removable chilled iron rings and weigh 5,000 lb.; a cross-beam keyed to the driving spindle carries a plow at either end which turns over the bricking mixture from both sides of the rollers into their path. The mold-disk is a cast-iron plate, 5 ft. in diameter and 2½ in. thick, having two rows of circular holes or molds (4 in. in diameter at the top and slightly larger at the bottom), pivoted at the center. Its movement is effected by a radial arm journaled at the center and attached at the outer end to a pitman connected with a crank-gear. The arm has a round steel pawl, which drops into one of the holes, and two plungers. As it is pulled forward, the pawl draws the disk around 15° and the pitman forces the two plungers into the molds, pushing out a pair of briquettes on a slow-moving endless belt. When the crank-gear again reaches its first position the pawl drops into the next hole and the plungers force out a second pair of briquettes and so on. The movement of rollers and disk is so adjusted that the rollers will travel 20 times over a pair of molds during their passage through the pan, thus firmly compressing the fluedust mixture.

The machine occupies a floor-space 9 ft. 6 in. by 11 ft. 6 in. and is 6 ft. 2 in. high; it weighs 27,000 lb., requires a 28 horse-power engine, and has a capacity of 4,200 bricks (4 by 2½ in.) per hour.

The machine requires two men at the pan, from two to four carriers to remove the bricks, and one foreman.

The green bricks received from a machine are loaded on brick trucks, and then piled up, rarely in racks, commonly on a covered floor, to a height of about 10 ft. and are air-dried for from two to three weeks. Drying chambers are of rare occurrence with Western smelting works, although the brick from them are much harder than if simply air-dried; with smelting works located in a less dry atmosphere the drying chambers may be necessary, especially during the cold season.

The burning of the brick is carried on in the ordinary up-draft kilns common for burning red-brick. The brick are

* Manufactured by the Chisholm, Boyd & White Co., Chicago, Ill.

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set up in arches about 30 courses high, the arches being 20 ft. long and the open portion about 9 courses high. Six or more arches are built side by side. A kiln 80 ft. long, 10 ft. wide, and 20 ft. high will hold about 100,000 brick. Fires are built in the arches, cordwood being commonly used. It takes from 3 to 4 cords to fire a kiln. The kiln begins to burn after a few hours of firing, which must then cease. It burns for 2 or 3 days and is pulled down after it has been given some time to cool.

The bricking of fluedust, which with hand-labor and a large percentage of lime cost formerly about $2 a ton, has been reduced by the use of machinery to from $0.20 to $0.25 a ton, excluding the cost of the lime.

Lime has probably to-day replaced all other binders. Harbordt* substituted clay, Church† used pan slimes, others have been successful with cement. Very satisfactory brick have lately been made with the waste molasses of sugar plantations. Its composition is shown by the following table‡ in percentages:

Name of Plantation.	Moisture.	Ash.	Sucrose. $C_{12}H_{22}O_{11}$.	Albuminoids.	Glucose. $C_6H_{12}O_6$.
Calumet.....	25.09	7.55	25.34	1.97	29.20
Calumet.....	26.15	9.35	26.02	2.40	28.93
Calumet.....	25.30	7.84	25.22	2.49	30.07
Calumet.....	25.09	7.01	25.46	2.30	31.31
Magnolia.....	30.37	9.34	27.65	1.22	21.12

The ash is principally lime and contains some potash. The fuel value according to Stubbs,§ is 37% of that of average Pennsylvania bituminous coal (H_2O 1.68, V. H-C. 28.43, F. C. 64.05, ash 5.83, with calorific power of 3,577 gr. cal. or 14,200 B. T. U.), 1 lb. molasses equaling about 0.37 lb. coal.

Previous to the improvements made in bricking, the common method of treatment was to melt the fluedust in a special reverberatory furnace or in the fuse-box of the roasting furnace, where it was added in quantities of, say, 100 lb. to the roasted ore. It was charged before the ore was drawn from the roasting-hearth, and was thus covered by the roasted ore, so that little metal was

* Private communication, 1891.

† *Engineering and Mining Journal*, Aug. 22, 1885; "Transactions of American Institute of Mining Engineers," xv., p. 611.

‡ "United States Department of Agriculture, Bulletin No. 18." tables 53 and 53, bis.

§ *The Louisiana Planter and Sugar Manufacturer*, July 18, 1896, p. 25.

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carried off mechanically. It helped to sinter the charge, but there was considerable loss in lead, although little in silver. Murray found that removing the finely divided carbon by washing assisted the fusion.

A charge of 150 tons of blast-furnace fluedust taken from the first 400 ft. of the dust-chamber of the Globe Smelting and Refining Co. gave, according to Iles:* SiO_2 16.6%, Fe 8.6%, CaO 5.2%, Pb 27.6%, Zn 3.0%., S 6.3%, Ag 20.0 oz., and Au 0.16 oz. After fusing it had changed to SiO_2 21.0%, Fe 12.4%, CaO 5.2%, Pb 37.8%, Zn 3.2%, S 6.4%, Ag 34.0 oz., and Au 0. 26 oz.

In melting fluedust in the reverberatory furnace, beside the slag are often formed some base bullion and matte. The three products of a charge gave Iles the following percentages:

Product.	Ag, Oz.	Au, Oz.	Pb.	SiO_2.	Fe.	CaO.	S.	Cu.	As.
Slag.....	21.0	0.08	18.4	22.4	12.4	2.0	1.6	.....	.....
Matte.....	160.7	1.54	46.8	.....	.....	.....	.....	15.4	1.4
Base bullion.....	450.9	6.12	.....	.....	.....	.....	.....	.....	.....

§ 95. Losses in Smelting.—The losses in smelting are due to slagging or to particles of metal being carried off in by-products and not recovered. As has already been stated (§ 68), ore-slags should not contain over 0.75% lead and ½ oz. silver to the ton with 300 oz. bullion; but they often contain over 1% lead and about 1 oz. silver, and are considered satisfactory. Special causes, such as the presence of foreign substances having a deleterious effect (§ 69), may make slags run still higher. One of the main sources of loss is that matte remains dissolved in the slag; often, also, pellets of matte do not settle out completely. This, according to Hering,† is sometimes due to the oxidizing action that certain slags have, especially if they contain magnetic oxide of iron (§ 68). These acting upon the sulphides set free sulphur dioxide, which adheres to pellets of matte and hinders them from settling.

The quantity of by-products (speise, matte, fluedust) formed has an important influence on the output of lead and silver, as they have to be roasted and resmelted several times, and each of these operations causes an unavoidable loss in metal. It is diffi-

* Private communication, 1897.

† *Oesterreichische Zeitschrift für Berg- und Hütten-Wesen*, 1898, p. 238.

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cult, therefore, to give an average figure of the lead and silver recovered in smelting in the blast furnace. It may be said that a total loss of from 6 to 7% of lead represents very good work, the lead assays being made in the dry way. As long as the loss does not exceed 10%, the work may be considered satisfactory. With silver the output is generally from 98 to 99%, the silver being also determined in the dry way. The yield in gold is 100%, the gold being paid to 0.05 oz. per ton; often there is a *plus* gold.

§ 96. Cost of Smelting.—The cost of smelting a self-fluxing ore at the large smelters of Denver and Pueblo is assumed to be $4 per ton, as previously stated (§ 36). The figure includes all incidental expenses; in fact, is the total cost.

The following estimate of the cost of smelting alone is for a single furnace, 42 by 120 in., based on Colorado data, and assumes that the furnace puts through 80 tons of ore in 24 hours and requires 20 tons of flux.

MATERIALS.

20 tons flux at $2	$40.00
15 tons coke (15%) at $5	75.00
5 tons coal at $3	15.00
Supplies (steel, oil, waste, etc.)	6.00

PAY-ROLL.

1 superintendent	$10.00	
1 assayer and chemist	4.50	
2 foremen at $4	8.00	
2 feeders at $2.70	5.40	
2 charge-weighers at $2.25	4.50	
8 charge-wheelers at $1.80	14.40	
2 furnace-men at $2.70	5.40	
4 furnace-helpers at $1.80	7.20	
5 laborers at $1.60	8.00	
1 bullion sampler	2.00	
1 carter and horse	2.75	
2 engine-men at $3	6.00	
2 firemen at $2	4.00	
1 blacksmith	3.00	
		$85.15
		$221.15
For unseen expense, 5%		
Delays in repairing furnace, 5%		$22.11
Cost of smelting 100-ton charge		
Cost of smelting 80-tons ore		$243.26
Cost of smelting one ton of ore		$3.04

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PART III.

DESILVERIZATION OF BASE BULLION.

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PART III.

DESILVERIZATION OF BASE BULLION.

§ 97. Introductory.—The final separation of silver and lead is universally accomplished by the process of cupellation. Keith*† desilverized base bullion on a working scale by means of electricity, but did not make a pecuniary success of it. Borchers‡ separated silver and bismuth from lead by a dry electrolytic process using alkali and alkali-earth chlorides as electrolyte. Roesing§ experimented at Tarnowitz, Silesia, in oxidizing lead in a Bessemer converter lined with basic refractory material. He worked with charges weighing 13,200 lb., and enriched lead from 12.4 to 196.3 oz. silver per ton, the fumes collected assaying 75% lead and 2.5% silver per ton. He also refined in a few minutes desilverized zinc-bearing lead. His process is being used for the production of litharge from refined lead. In what way this new idea can be applied to desilverization so as to compete with the present methods remains to be seen. Formerly all argentiferous lead was cupelled, but this was found to have disadvantages, prominent among which are the cost and the loss in metal, the limit being very soon reached when the separation of silver from lead ceases to pay. This is the case with base bullion assaying about 30 oz. silver to the ton. Below this, the silver recovered will hardly pay for the labor, fuel, and material required, the loss in metal, and the impurity of the lead obtained from the reduction of the litharge. It becomes, therefore, neces-

*_Engineering and Mining Journal_, July 13, 27, 1878; June 3, 1882; Dec. 15, 1883, p. 372; "Transactions of American Institute Mining Engineers," x., p. 312; xiii., p. 310.

†Hampe's criticism; _Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen_, xxx., p. 91; _Engineering and Mining Journal_, March 18, 1882.

‡ _Zeitschrift für Electrotechnik und Electrochemie_, i., p. 13; "The Mineral Industry," iii., p. 442.

§ _Revue universelle des mines_, 1892, xvii., p. 110; _Iron_, March 25, 1892; _Berg- und Hüttenmännische Zeitung_, 1892, p. 102; _Engineering and Mining Journal_, April 16, 1892; _Stahl und Eisen_, 1892, p. 370.

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sary to concentrate the silver in a smaller amount of lead before cupelling. The processes of Pattinson and Parkes do this successfully.

The progress made in desilverizing during the last 60 years is well illustrated by the following table. The figures published by Hermann* are derived from the actual working results obtained at the Ems Smelting and Refining Works, Prussia, where the three processes were used one after another. To these have been added the amount of lead to be cupelled and the traces of gold.

	Cupellation Process.	Pattinson's Process.	Parkes' Process.
Cost	3	1½	1
Amount of lead to be cupelled	100	13	5
Loss in lead and silver	6	2	1
Impurities remaining in the lead, per cent	0.2	0.05	0.015
Silver remaining in the lead, ounces per ton	0.73	0.58	0.17
Traces of gold	Lost.	Lost.	Recovered.

The task of modern desilverizing works (or refining works, as they are also called) is not only to separate effectively and cheaply the precious metals from lead, but also to make out of a base bullion containing from 95 to 98% of lead a refined lead of not less than 99.9% of lead and salable products of the impurities contained in it, such as copper, tin, arsenic, and antimony.

In the following chapters, cupellation, being now only an auxiliary to Pattinson's and Parkes' processes in desilverizing base bullion, although still an independent process in the manufacture of litharge, will be treated last. The three processes will be discussed in the following order:

Pattinson's Process.
Parkes' Process.
Cupellation.

* *Berg- und Hüttenmännische Zeitung*, 1883, p. 382.

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CHAPTER IX.

PATTINSON'S PROCESS.

§ 98. Introductory Remarks.—The process is based on the fact discovered by Pattinson in 1833, that if silver-bearing lead is melted and cooled down almost to its fusing-point, crystals of lead will separate which are much poorer in silver than the original lead. If they are removed and the process is repeated, always adding fresh lead of the same tenor in silver, a large quantity of market lead low in silver will result, and a small amount of enriched lead ready to be cupelled. By the repeated meltings and crystallizations many of the impurities will also have been collected in drosses, and the market lead become purified.

Pattinson* observed that on heating carefully a bar of lead that ran low in silver, until a few drops of metal oozed out, these were richer in silver than the residual lead, while with a bar running high in silver just the reverse was the case. These phenomena, which formerly were not understood, became clear with the improved knowledge of the laws of solution.† A low-grade fused alloy of lead and silver is to be considered as a dilute solution of silver in lead. Upon cooling, the pure solvent, in this case the lead, will, as usual, separate from it, the melting point of the solution being lower than that of the pure solvent. With a high-grade, we have a solution of lead in silver and upon cooling, the silver will separate out.

Reich‡ measured the melting points of different lead-silver alloys, and found that lead with 1.89 oz. silver per ton melted at 321° C.; with 139.03 oz., at 309°; and that with 656.23 oz. silver the lowest melting-point was reached. If, however, the lead

* Percy, "Metallurgy of Lead," London, 1870, p. 137.
† See also Kossman, *Stahl und Eisen*, 1893, p. 606.
‡ *Berg- und Hüttenmännische Zeitung*, 1862, p. 251.

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contained much silver, for instance 33 or 50%, its melting point was much higher than that of lead free from silver. Tunner* determined the melting points of the following alloys:

Degr es Centigrade.	Parts Silver.	Parts Lead.
472	2.0	8.0
506	2.5	7.5
540	3.0	7.0
575	3.5	6.5
609	4.0	6.0

Behrens† examined microscopically alloys of lead with 0.5, 1.0, 1.5 and 2.5% silver. The inferences to be drawn from Reich's figures are that Pattinson's process is adapted only for low-grade bullion, and that the enriching of the liquid lead can be carried only to a certain degree. The following table, by Reich,‡ shows how far the silver can be concentrated in the lead:

Ounces Silver per Ton.		
In the Molten Lead Before Crystallization.	In the Crystals.	In the Liquid Lead.
205.83	113.74–135.91	298.95
213.49	92.75–109.06	313.88
281.84	119.58–198.33	422.91
288.16	113.74–181.99	446.24
420.57	198.91	560.57
609.57	586.53	659.15
615.15	503.99–646.81	655.65
643.40	645.15	660.32

The process has to stop when the liquid lead assays from 600 to 650 oz. silver per ton. In practice the concentration is stopped when the liquid lead assays from 450 to 500 oz., as the nearer the silver contents approach the 650 oz. silver per ton, the smaller become the crystals, and the more difficult is it to drain off the liquid lead, especially as this also tends to solidify at the same time.

The process of concentrating the silver in a small amount of lead may be conducted according to two systems, called the method by thirds and the method by eighths. In the first of these systems two-thirds of the lead contained in the kettle is withdrawn in the

* *Oesterreichische Zeischrift für Berg- und Hütten-Wesen*, 1892, p. 80.
† *Das mikroskopische Gefüge der Metalle und Legirungen*, Hamburg-Leipsic, 1894, p. 47.
‡ *Berg- und Hüttenmännische Zeitung*, 1892, p. 251.

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form of crystals, while one-third remains behind as liquid lead. The crystals will then be about half as rich, and the liquid lead about twice as rich, as the original bullion. In the second system the bullion in the kettle is divided into seven-eighths crystals and one-eighth liquid lead, and the silver contents of the crystals is approximately one-third as much, and of the liquid lead approximately three times as much, as that of the original bullion. The latter method is, therefore, to be applied to low-grade bullion. Stetefeldt* tried to find a general mathematical formula which would show the proportions in which leads of different contents in silver should be divided to attain, with as few crystallizations as possible, a market lead of a certain tenor in silver and an enriched lead. In practice the two methods by thirds and by eighths have become standards, especially the former. A variation of the method by thirds, the one with intermediary crystals,† aims to reduce the number of crystallizations. The contents of the kettle are divided—two-thirds crystals and one-third liquid lead—and the crystals taken out. The liquid lead, however, undergoes directly a second crystallization, so that intermediary crystals and final liquid lead will result. Thus the original lead is divided into $\frac{2}{3}=\frac{4}{6}$ normal crystals, assaying one-half as much as the original lead; $\frac{2}{6}$ intermediary crystals, assaying the same as the original lead; and $\frac{1}{6}$ liquid lead, four times as rich as the original lead. The method has, however, been abandoned where it was tried, as it complicated the process, and as considerable amounts of slightly enriched leads had to be kept on hand.

To carry out Pattinson's process successfully the base bullion must not be very impure or run too high in silver, and a sufficient amount must be used to permit the necessary careful regulation of temperature. All the foreign metals contained in the lead interfere with the crystallization and the effectual separation of the liquid lead from the crystals. Ordinary lead can be sufficiently purified by poling (§ 116) and removing the dross that collects on the surface; if tin, arsenic, and antimony are present to any extent the lead has to be softened (§ 104) at a bright-red heat before the crystallization can proceed. Of the metals‡ com-

* *Berg- und Hüttenmännische Zeitung*, 1863, pp. 64, 69, 77.
† Stetefeldt, *Ibid.*, 1868, pp. 297, 881.
‡ *Berg- und Hüttenmännische Zeitung*, 1889, p. 116.

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monly found in base bullion, antimony, bismuth, and nickel are concentrated in the liquid lead; arsenic in the crystals; copper that has not been removed with the dross remains equally distributed in both products. The concentration of bismuth in the liquid lead is of special importance, as the crystallized lead, if it is to be used for corroding purposes, may contain only very small amounts of bismuth (§ 11).

It has been found* that the richer the lead is in silver the more easily will the bismuth be concentrated in it. Thus, base bullion with 145.83 oz. silver per ton and 0.06% bismuth gave rich lead with 583.32 oz. silver per ton and 0.18% bismuth, and market lead with 0.44 oz. silver per ton and 0.02% bismuth. When the base bullion was richer in bismuth, the silver remaining the same, the rich lead retained more bismuth (0.29%), and this was also the case with the market lead (0.054%).

How slowly the bismuth contents of the crystallized lead diminish in Pattinsonizing, if compared with the silver, is shown in the subjoined diagram (Fig. 321).

The curve *A B* refers to a base bullion containing 0.1% bismuth, the curve *C D* to one with 0.25%. The decrease of bismuth in a series of crystallizations carried out with the method by thirds is shown in the following table; the figures indicate the percentages of bismuth:

No. of Kettle.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.	13.	14.
Lead before crystallizing	0.242	0.205	0.200	0.208	0.183	0.173	0.162	0.137	0.132	0.110	0.105	0.088	0.064	0.053
Res'lting crystals.....	0.235	0.207	0.204	0.208	0.176	0.158	0.157	0.128	0.115	0.112	0.085	0.074	0.070	*a*0.054
Resulting liquid lead....	0.290	0.247	0.243	0.242	0.181	0.217	0.212	0.183	0.174	0.140	0.130	0.101	0.093	0.065

(*a*) Market lead.

The kettles used with the Pattinson process hold from 6 to 15 tons of lead, and the smallest permissible quantity is 2½ tons. Either the crystals are taken out with a perforated ladle or the liquid lead is drawn off.

§ 99. Description of Plant and Mode of Conducting the Process.—The plant consists of a set of from 8 to 15 spherical kettles (with the method by thirds), built closely together in a

* Junge, *Freiberger Jahrbuch*, 1895, p. 3; "The Mineral Industry," v., p. 898.

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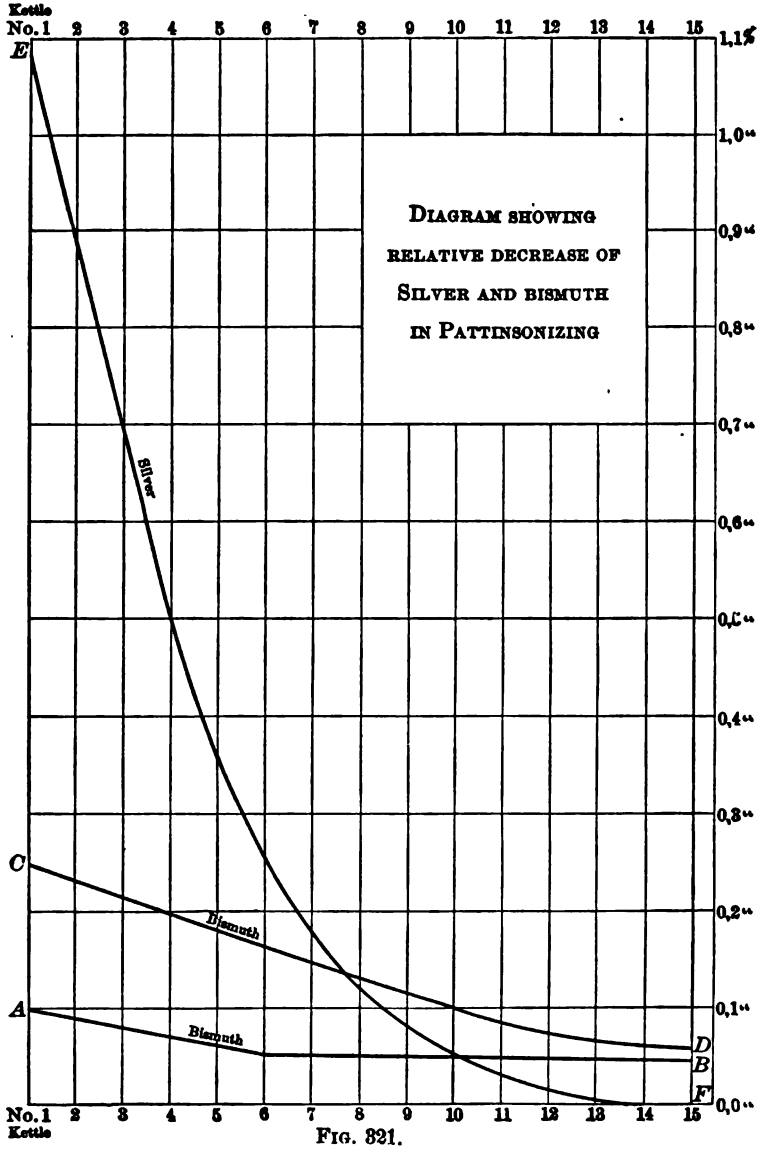


Fig. 321.

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row. Each kettle has a separate fireplace, so constructed that the flame shall pass beneath and behind the kettle, thence into a flue encircling it, and finally into the chimney, which has a damper to regulate the draft. The details of the construction are the same as with the desilverizing-kettle of Parkes' process (§ 108).

The mode of operation in outline with the method by thirds is as follows: In the central kettle the base bullion is melted down, drossed and, if necessary, poled. The fire below is then withdrawn and transferred to a neighboring kettle. The cooling is promoted by sprinkling water on the surface from a rose. Crusts adhering to the sides of the kettle are pushed down into the lead, where they melt again. This is the work of one man, who also stirs the metal continuously until the smooth surface becomes rough with crystals. His partner now inserts at the rim of the kettle a long-handled skimmer that has been warmed, and works it across the bottom of the kettle to the opposite side, then back to the middle, where, after jerking it to remove the liquid lead, he discharges the dry crystals into the neighboring kettle, generally the one to the right ("down the house"). The operation is continued until two-thirds of the contents of the kettle have been removed in the form of crystals. The liquid lead is then ladled into the kettle on the left ("up the house"). To the kettle at the right, being two-thirds full of crystals, one-third of lead of the same tenor is added, and the kettle at the left, being one-third full of liquid lead, is filled with a corresponding amount ($\frac{2}{3}$) of lead of its tenor. The kettles are heated, and the cooling, crystallizing, and ladling carried on in the same way as in the original bullion kettle. This becomes again filled from the crystals of the kettle on the left and the liquid lead of that on the right. Thus the operations are continued, the lead of the kettles to the right decreasing in tenor till that of the last one, the market-pot, assays from 0.3 to 0.5 oz. silver per ton; that to the left increasing till the maximum of 650 oz. is reached.

From the foregoing it will be seen that before the whole plant can be in working order quite a number of crystallizations have to be carried on, so as to have on hand the necessary amounts of lead of different silver contents required to fill the kettles.

Pattinson's process in its original form is still in use in England, Freiberg, and perhaps some other places. As it is improb-

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able that it will be introduced anywhere in the United States, this general outline will suffice. Full details are given in the works of Percy,* Kerl,† Schnabel,‡ Stölzel,§ Roswag,‖ Grüner,¶ and the paper by Teichmann.**

In order to reduce the hard work necessary in withdrawing the crystals and ladling out the lead, as well as to insure a more regular crystallization and better separation of crystals and liquid lead, machinery was introduced into Pattinson's process; but the main modification of the original process, which has become the standard one of to-day, is that by Luce and Rozan, who stir by steam and draw the liquid lead off, leaving the crystals in the kettle.

§ 100. Luce and Rozan's†† Process (Steam Pattinson Process).—The advantages of the steam are, that it causes a regular crystallization and a good separation of the lead from the crystals, and that it poles the lead, which being much exposed to the influence of the air becomes purified. Thus moderately pure base bullion, containing from 0.5 to 0.75 per cent. of foreign metals, can be desilverized without previous softening. It is claimed that lead with a little antimony and copper is even preferable, as less dross forms than would be the case if it were free from these metals. Of course, lead containing appreciable amounts of arsenic and antimony has to be softened with this process, as with any other, before it can be satisfactorily desilverized.

The way in which the process is carried out at Pribram,‡‡ Bohemia, may serve as an example.

Figs. 322, 323 and 325 show the general arrangement of the plant, consisting of two melting-pans *a*, one crystallizing-pot *l*, and two large conical molds. The steam crane is not shown; it is placed on the side of the crystallizing-pot, and serves to transfer the cakes of lead from the molds to the storage-place, and thence to the melting-pans, and to tip the latter. The trough-shaped cast-

* "Metallurgy of Lead," London, 1870, p. 121.

† "Grundriss der Metallhüttenkunde," Leipsic, 1881, p. 225.

‡ Metallhüttenkunde, Berlin, 1894, vol. i., p. 511.

§ "Metallurgie," Brunswick, 1863-1886, p. 1122.

‖ "La désargentation de plomb," Paris, 1884, pp. 211 and 267.

¶ *Annales des mines*, 1868, xiii., p. 379.

** *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xv., p. 40.

†† Luce and Rozan, *Annales des mines*, 1873, iii., p. 160; Cookson, *Iron*, Sept. 22, 1881; *Engineering and Mining Journal*, Oct. 8, 1881.

‡‡ Zdráhal, *Oesterreichisches Jahrbuch*, xxxviii., p. 1; private notes, 1890, and correspondence, 1897.

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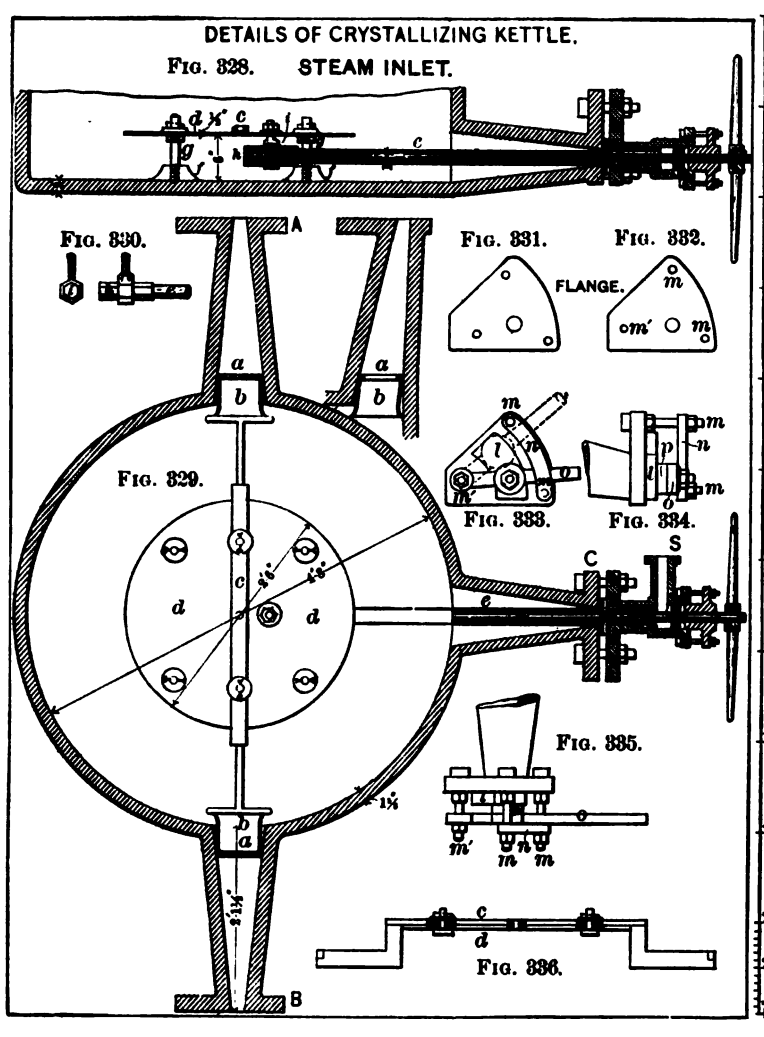
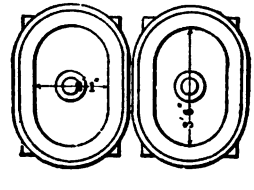


FIG. 337.



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iron melting-pans *a*, each holding 15,430 lb. of lead, are placed behind, and 2 ft. 4 in. above, the top of the crystallizer *l*. They rest with their rims on the cast-iron frame *b*, and are emptied by tipping, by means of the crane, over the inclined plate *c*, which discharges the lead through the stationary cast-iron trough *d*, and a movable sheet-iron trough (not shown), into the crystallizer *l*. Each pan has its separate fireplace (Figs. 322 and 327) on the side, from which the gases, after passing upward (Fig. 322) through a long flue, surround the bottom of the pan, and descend either directly through flue *e* (Figs. 322 and 323) to the chimney, or first encircle the upper part of the crystallizer (*l*, Figs. 322 and 323), and then pass off through the flue *f* (Fig. 323); the passage of the gases is regulated by the dampers *g* (Fig. 322) and *h* (Fig. 323). On the oval hearth (Fig. 325) are built two small walls *k* (Figs. 322, 323, 325), in order that the flame may pass close to the pan. Any lead coming from a leaking pan collects in the lowest part of the hearth (Fig. 322), tamped with brasque, and is discharged outside of the brickwork.

The crystallizer *l* (Figs. 322 and 323) is a flat-bottomed cylindrical pot holding 44,100 lb. of lead, or nearly three times as much as one melting-pan. It has (Fig. 329) near the bottom two spouts, *A* and *B*, closed by slide-valves (Figs. 333, 334, 335), for discharging the lead. At a right angle to the plane of these spouts is the steam-inlet *C* (Figs. 328 and 329). The pot rests (Figs. 322 and 327) on the cast-iron frame *p*, supported by four cast-iron pillars *o*. The top of the crystallizer (Fig. 322) is covered with a conical hood, ending in a sheet-iron pipe, through which steam and dust are carried off to be condensed. The hood has three openings closed by doors—one at the front above the steam-inlet, and two at the sides above the lead-spouts; and a small hole near the top, for the water inlet-pipe. The crystallizer is fired (Figs. 322 and 327) from the passage below the melting-pan *a*; the flames pass along the bottom, turn to the left, and encircle the lower part of the pot; they are checked by being forced to make their way through the narrow passage *q* before passing downward and off through the flue *r*. On either side of the large central fireplace is (Figs. 326 and 327) a smaller one, which serves to heat the discharge-spouts *m* before using them. Each of the discharge-spouts *A* and *B* (Fig. 329) has a

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perforated cast-iron straining-plate to keep back the crystals when the liquid lead is being run off. These are held in place by wrought-iron arms *b* and the cast-iron frame *c* (Figs. 329 and 336), which is fastened by key-bolts to the baffle-plate *d*. The spouts are closed by a slide-valve (Figs. 333, 334, 335). To the flange of the spout (Fig. 331) is fastened with countersunk screws and a red-lead cement, a plate (Fig. 332) of the same form, with one face planed smooth, having four openings to correspond to those of the flange; *i.e.*, the central lead-discharge and the three holes near the rounded corners. Through these the bolts *m'* and *m* are passed, *m'* serving as a pivot for the lever *o* and *m* to tighten the guide *n*. To the lever is fastened the plate *p*, also having one planed face. In Fig. 333 is shown the position of the lever when the discharge is closed. In order to open it the nuts of the screw-bolts *m* are loosened, and the lever pushed into the second position, shown by the dotted lines. The lead from the crystallizer passes through the two lead spouts into two tapering molds (Figs. 325 and 337), each of which holds about 6,610 lb. of lead. The steam-inlet consists of the following parts (Fig. 328). On the flat bottom of the crystallizer are four bosses *f* into which fit the screws *g*. On the collar of these is placed and keyed the cast-iron circular baffle-plate *d*, with its small opening *c* in the center. It serves to distribute the steam evenly, and to make it rise regularly in the pot. From it is suspended by an eye-bolt *i* with hexagonal eye, the nozzle *h*, into which is screwed the steam-pipe *e*. Through it passes the rod *k*, moved to and fro at one end by the thread and cross-bar; the other end, which is conical, fits into the conical valve-seat of the nozzle *h*, and closes or opens the steam-outlet. The steam entering at *S* (Fig. 329) passes through the small annular space between pipe and rod, and out at *h*, when the valve is open.

The method of working is the one by thirds. The mode of conducting the process is simple. Suppose the process to be going on at the stage when the liquid lead has been drained off from the crystallizer; the valves have been again closed and the crystals liquefied. One pan will be full of liquid lead of the same tenor in silver as the melted crystals to be discharged into the crystallizing pot, while the other will contain two cakes of lead that are being melted down. They will have the same silver contents as the crystals remaining in the crystal-

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lizer after the operation to be described has taken place. The melting down of two cakes takes about six hours.

The lead from the pan is run out by inserting two hooks, fastened to the chain suspended from the pulley of the crane, into the ears of the pan, and raising it slowly. After the lead has been discharged into the crystallizer the doors of the hood are closed and steam introduced for two minutes, and shut off again when the pulverulent dross which has collected on the surface is removed. Now the crystallization proper begins. The fire below the crystallizer is withdrawn and divided between the two small fireplaces on either side, from which the lead-spouts are warmed. Steam is turned on, and a small jet of water is allowed to play at short intervals upon the surface of the lead. Every time the water is let on there are small explosions, and as soon as they become too violent the water is shut off again, while the steam enters continuously. The steam has 45 lb. pressure to the square inch, and care must be taken to have it dry. About 15 minutes after introducing the steam, the lead that has been splashed up on the upper edge of the pot, or on the hood, has to be removed. The steam is shut off, the doors in the hood are opened one after the other, and the solidified lead-crusts broken off with a chisel-pointed bar, and pushed back into the lead. This shutting off of steam to remove the lead is repeated at least twice. While the crystallization is going on, the two cakes of lead required to fill again the melting-pan, just emptied into the crystallizer, are hoisted from below with the crane, and deposited one on top of the other in the pan. The crystallization is finished when the normal amount of steam can no longer overcome the resistance offered by the crystals. The result is that the boiling ceases, and the surface of the crystalline mass of lead shows only a slow, wave-like motion. Two-thirds of the original lead have now been converted into crystals, the lead having assumed a mushy consistency. Water and steam are shut off, the slide-valves are opened, and the liquid lead is discharged into the molds, which takes from eight to ten minutes. In these have previously been placed iron hooks, by which the cakes of lead, when cold, may be lifted out. The whole process of crystallization lasts about one hour. While the lead is running off, the fire from the two small fireplaces is returned to the grate below the crystallizer, and urged in preparation for the next operation.

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The melting and other work require three hours, so that one operation lasts four hours.

Eleven crystallizations are necessary to obtain market-lead from liquated base bullion averaging 146.12 oz. silver to the ton. The following table shows the average assay-value in ounces per ton of the different leads produced during a whole year's work. The second column represents the results of the same process at Eureka,* Nev.

Pribram.		Eureka.
Market Lead,	0.43	1.25 Market Lead.
I.,	0.87	2.5
II.,	1.75	5.0
III.,	3.21	9.0
IV.,	6.41	18.0
V.,	10.21	30.0
VI.,	18.96	50.0
VII.,	29.16	75.0
VIII.,	40.83	100.0
IX.,	55.41	150.0
X.,	98.33	460.0 Rich Lead.
XI. Blast Furnace Bullion,	142.91	
XII., Rich Lead,	262.49	

Six charges are run in 24 hours; two men working as partners attend to the crystallization, all the handling of the lead being done by the engineer and his helper. The products of the process are rich lead, desilverized lead, dross, and fluedust. The rich lead is cupelled, the desilverized lead is refined in a reverberatory furnace and molded into market lead, the dross and fluedust are worked with similar products from other parts of the works. The output of metal is shown by the following table:

Recovered in	From 100 Pounds Base Bullion Charged.	From 100 Ounces Silver Charged.	From 100 Pounds Lead Charged.
Rich lead	42.99	97.36	42.58
Desilverized lead	44.76	0.17	45.01
Scrap lead	3.02	0.61	3.06
Dross	9.94	1.45	8.04
Fluedust	0.46	0.07	0.35
Loss		0.34	0.99
Total	101.17	100.00	100.00

The material consumed for desilverizing 100 tons of base bullion excluding the refining of the lead, is: Charcoal, 25 bushels;

* Curtis, "Silver-Lead Deposits of Eureka, Nev.," monograph vii., U. S. Geological Survey, 1884, p. 163.

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bituminous coal, 26.60 tons (for melting and desilverizing); bituminous coal, 8.23 tons (for raising steam).

Balling* gives the life of a melting pan as 40 days, of a crystallizer as 120 days, which seems very low.

It has already been stated that before the process can be carried on normally, a number of preliminary crystallizations have to take place to furnish the necessary intermediary products. On account of the possibility, perhaps the probability, that the process will be combined with the Parkes process in this country because of the behavior of bismuth, full details of the manner of starting the plant at Pribram are given, although they involve much repetition.

The outline represents 11 *operations*, each requiring one more *charge* than the one preceding, before market lead can be produced, and gives the 12 grades of argentiferous lead, I.—XII., the values of which are given on page 422. The "quintals" used at Pribram have been retained, as changing them into pounds would have deprived the scheme, complicated as it is, of the necessary clearness. One quintal is equal to 100 kilograms and these equal 220.46 lb. avoirdupois.

The start is of course made with blast furnace bullion, B. F. XI, assaying 142.91 oz. per ton, and 180 quintals is the amount given below in Operation 1, which upon crystallization would furnish 60 quintals liquid lead, L. L. XII, with 262.49 oz. silver per ton and 120 quintals crystallized lead, C. L. X., with 93.33 oz. silver per ton. As the separation according to thirds is never accurate, these figures are not actually obtained, and as further, in melting, poling, etc., a considerable amount of dross is formed, the start is made with 190 quintals instead of with 180 quintals, and in the operations 130 quintals are charged instead of 120. Even with these larger amounts, the weight of the market lead obtained ranges only from 105 to 112 quintals.

OPERATION I.

Charge 1. Blast furnace bullion B.F. 180, XI.; gives 60 L. L., XII., which go to the cupelling furnace (cupel) and 120 C. L., X., which is liquefied in the crystallizer, drawn off into the molds, and stacked on the floor (floor).

* "Metallhüttenkunde," p. 295.

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TABLE OF PRELIMINARY OPERATIONS OF THE LUCE-ROZAN PROCESS.

Operation No.	Charge						Gives Liquid Lead.			Gives Crystallized Lead.			
	Charge No.	Quintals.	Kind.	Source. Op.	Source. Ch. No.	Total Quintals.	Quintals.	Kind.	Destination.	Quintals.	Kind.	Destination. Operation No.	Destination. Charge No.
1	1	180	xi	Bl.	F.	180	60	xii	Cupel.	120	x	Floor.	
2	1	180	xi	Bl.	F.	180	60	xii	Cupel.	120	x	2	2
2	2	120	x	2	1	180	60	xi	Floor.	120	ix	Floor.	
2		60	x	1	1								
3	1	120	xi	Bl.	F.	180	60	xii	Cupel.	120	x	3	2
3		60	xi	2	2								
3	2	120	x	3	1	180	60	xi	Floor.	120	ix	3	3
3		60	x	1	1								
3	3	120	ix	3	2	180	60	x	Floor.	120	viii	Floor.	
3		60	ix	2	2								
4	1	120	xi	Bl.	F.	180	60	xii	Cupel.	120	x	4	2
4		60	xi	3	2								
4	2	120	x	4	1	180	60	xi	Floor.	120	ix	4	3
4		60	x	3	3								
4	3	120	ix	3	2	180	60	x	Floor.	120	viii	4	4
4		60	ix	2	2								
4	4	120	viii	4	3	180	60	ix	Floor.	120	vii	Floor.	
4		60	viii	3	3								
5	1	120	xi	Bl.	F.	180	60	xii	Cupel.	120	x	5	2
5		60	xi	4	4								
5	2	120	x	5	1	180	60	xi	Floor.	120	ix	5	3
5		60	x	4	3								
5	3	120	ix	5	2	180	60	x	Floor.	120	viii	5	4
5		60	ix	4	4								
5	4	120	viii	5	3	180	60	ix	Floor.	120	vii	5	5
5		60	viii	3	3								
5	5	120	vii	5	4	180	60	viii	Floor.	120	vi	Floor,	
5		60	vii	4	4								
6	1	120	xi	Bl.	F.	180	60	xii	Cupel.	120	x	6	2
6		60	xi	5	2								
6	2	120	x	6	1	180	60	xi	Floor.	120	ix	6	3
6		60	x	5	3								
6	3	120	ix	6	2	180	60	x	Floor.	120	viii	6	4
6		60	ix	5	4								
6	4	120	viii	6	3	180	60	ix	Floor.	120	vii	6	5
6		60	viii	5	5								
6	5	120	vii	6	4	180	60	viii	Floor.	120	vi	6	6
6		60	vii	4	4								
6	6	120	vi	6	5	180	60	vii	Floor.	120	v	Floor.	
6		60	vi	5	5								
7	1	120	xi	Bl.	F.	180	60	xii	Cupel.	120	x	7	2
7		60	xi	6	2								
7	2	120	x	7	1	180	60	xi	Floor.	120	ix	7	3
7		60	x	6	3								
7	3	120	ix	7	2	180	60	x	Floor.	120	viii	7	4
7		60	ix	6	4								
7	4	120	viii	7	3	180	60	ix	Floor.	120	vii	7	5
7		60	viii	6	5								
7	5	120	vii	7	4	180	60	viii	Floor.	120	vi	7	6
7		60	vii	6	6								
7	6	120	vi	7	5	180	60	vii	Floor.	120	v	7	7
7		60	vi	5	5								
7	7	120	v	7	6	180	60	vi	Floor.	120	iv	Floor.	
7		60	v	6	6								
8	1	120	xi	Bl.	F.	180	60	xii	Cupel.	120	x	8	2
8		60	xi	7	2								
8	2	120	x	8	1	180	60	xi	Floor.	120	ix	8	3
8		60	x	7	3								
8	3	120	ix	8	2	180	60	x	Floor.	120	viii	8	4
8		60	ix	7	4								
8	4	120	viii	8	3	180	60	ix	Floor.	120	vii	8	5
8		60	viii	7	5								
8	5	120	vii	8	4	180	60	viii	Floor.	120	vi	8	6
8		60	vii	7	6								
8	6	120	vi	8	5	180	60	vii	Floor.	120	v	8	7
8		60	vi	7	7								

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TABLE OF PRELIMINARY OPERATIONS OF THE LUCE-ROZAN PROCESS.—(*Con.*)

Operation No.	Charge.						Gives Liquid Lead.			Gives Crystallized Lead			
	Charge No.	Quintals.	Kind.	Source.		Total Quintals.	Quintals.	Kind.	Destination.	Quintals.	Kind.	Destination.	
				Op. No.	Ch. No.							Operation No.	Charge No.
8	7	120	v	8	6	180	60	vi	Floor.	120	iv	8	8
8		60	v	6	6								
8	8	120	iv	8	7	180	60	v	Floor.	120	iii	Floor.	
8		60	iv	7	7								
9	1	120	xi	Bl.	F.	180	60	xii	Cupel.	120	x	9	2
9		60	xi	8	2								
9	2	120	x	9	1	180	60	xi	Floor.	120	ix	9	3
9		60	x	8	3								
9	3	120	ix	9	2	180	60	x	Floor.	120	viii	9	4
9		60	ix	8	4								
9	4	120	viii	9	3	180	60	ix	Floor.	120	vii	9	5
9		60	viii	8	5								
9	5	120	vii	9	4	180	60	viii	Floor.	120	vi	9	6
9		60	vii	8	6								
9	6	120	vi	9	5	180	60	vii	Floor.	120	v	9	7
9		60	vi	8	7								
9	7	120	v	9	6	180	60	vi	Floor.	120	iv	9	8
9		60	v	8	8								
9	8	120	iv	9	7	180	60	v	Floor.	120	iii	9	9
9		60	iv	7	7								
9	9	120	iii	9	8	180	60	iv	Floor.	120	ii	Floor.	
9		60	iii	8	8								
10	1	120	xi	Bl.	F.	180	60	xii	Cupel.	120	x	10	2
10		60	xi	9	2								
10	2	120	x	10	1	180	60	xi	Floor.	120	ix	10	3
10		60	x	9	3								
10	3	120	ix	10	2	180	60	x	Floor.	120	viii	10	4
10		60	ix	9	4								
10	4	120	viii	10	3	180	60	ix	Floor.	120	vii	10	5
10		60	viii	9	5								
10	5	120	vii	10	4	180	60	viii	Floor.	120	vi	10	6
10		60	vii	9	6								
10	6	120	vi	10	5	180	60	vii	Floor.	120	v	10	7
10		60	vi	9	7								
10	7	120	v	10	6	180	60	vi	Floor.	120	iv	10	8
10		60	v	9	8								
10	8	120	iv	10	7	180	60	v	Floor.	120	iii	10	9
10		60	iv	9	9								
10	9	120	iii	10	8	180	60	iv	Floor.	120	ii	10	10
10		60	iii	8	8								
10	10	120	ii	10	9	180	60	iii	Floor.	120	i	Floor.	
10		60	ii	9	9								
11	1	120	xi	Bl.	F.	180	60	xii	Cupel.	120	x	11	2
11		60	xi	10	2								
11	2	120	x	11	1	180	60	xi	Floor.	120	ix	11	3
11		60	x	10	3								
11	3	120	ix	11	2	180	60	x	Floor.	120	viii	11	4
11		60	ix	10	4								
11	4	120	viii	11	3	180	60	ix	Floor.	120	vii	11	5
11		60	viii	10	5								
11	5	120	vii	11	4	180	60	viii	Floor.	120	vi	11	6
11		60	vii	10	6								
11	6	120	vi	11	5	180	60	vii	Floor.	120	v	11	7
11		60	vi	10	7								
11	7	120	v	11	6	180	60	vi	Floor.	120	iv	11	8
11		60	v	10	8								
11	8	120	iv	11	7	180	60	v	Floor.	120	iii	11	9
11		60	iv	10	9								
11	9	120	iii	11	8	180	60	iv	Floor.	120	ii	11	10
11		60	iii	10	10								
11	10	120	ii	11	9	180	60	iii	Floor.	120	i	11	11
11		60	ii	9	9								
11	11	120	i	11	10	180	60	ii	Floor.	120	x	Market Lead.*	
11		60	i	10	10								

* After it has been refined in a reverberatory furnace to remove the As and Sb it still retains.

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OPERATION II.

Charge 1. B. F. 180, XI.; gives 60 L. L., XII., to cupel, and 120 C. L., X., remaining in the crystallizing kettle; there are added from the melting pots 60 C. L., X., from Operation I, charge 1, to make up

Charge 2. 180, X. gives 60 L. L., XI., to floor, and 120 C. L., IX., to floor.

OPERATION III.

Charge 1. B. F. 120, XI., and 60, XI., from Operation II. charge 2, gives 60 L. L., XII., to cupel, and 120 C. L., X; add remaining 60 C. L., X., from Operation I, charge 1, to make up.

Charge 2. 180, X.; gives 60 L. L., XI., to floor, and 120 C. L., IX.; add 60 C. L., IX., from Operation II., charge 2, to make up

Charge 3. 180, IX. gives 60 L. L., X., to floor and 120 C. L., VIII., to floor, etc.

The preceding table gives systematically arranged all the preliminary steps necessary to arrive at the point where the first market lead is produced.

It thus requires 66 crystallizations to obtain the intermediary products necessary for normal work. One-half of the crystallized lead from Operation X, Charge 10 remains over to be worked off subsequently.

In comparing the processes of Luce-Rozan and Pattinson, Cookson* comes to the conclusion that the former is to be preferred by far, as the softening of the lead is not so imperative, the cost of labor only 20%, and that of fuel 40%, of the cost by Pattinson's process; and, lastly, it produces only 33% of the amount of drosses obtained by Pattinson. The drawback of the greater original cost, and the continued expense of repair and renewal, is more than made up by the advantages.

* *Engineering and Mining Journal*, April 12, 1879.

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CHAPTER X.

PARKES' PROCESS.

§ 101. Introductory Remarks.—Parkes' process is based on the fact that if from 1 to 2% zinc is stirred into melted base bullion, it will deprive the latter of its silver, and form an alloy, which, being less fusible than lead, and having a lower specific gravity, will become hard and float on the surface of the lead, whence it can be removed and treated separately; while the lead which has taken up some zinc is refined, and is then ready for the market. Karsten discovered in 1842 that argentiferous lead could be desilverized by the use of zinc, but his discovery could be applied in practice only when Parkes found the means (1850–52) of working the zinc-silver-lead crust and refining the lead.

The theory on which the process is based has always been that silver has a greater affinity for zinc than for lead, and therefore combines with it when added to molten bullion. According to Roesing* the statement has to be modified by saying that, while silver has a greater affinity for zinc than for lead, it has less affinity for zinc-bearing lead than either for zinc or for pure lead, and that this is the main cause why argentiferous lead can be desilverized by means of zinc.

According to Alder-Wright and Thompson,† zinc and silver form two definite alloys, $AgZn_8$ and Ag_4Zn_5. The former shows under the microscope‡ dendritic forms and is an unstable compound. It has the property of dissolving lead, and is itself dissolved by lead to a greater extent than either pure zinc or the alloy Ag_4Zn_5. If it be kept molten for some time, holding its

* *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xxxvii., p. 76.

† Proceedings of the Royal Society, 1890, xlviii., p. 32; *Engineering and Mining Journal*, Dec. 20, 1890.

‡ Behrens, "Die mikroskopischen Gefüge der Metalle und Legirungen," Hamburg-Leipsic, 1894, p. 46.

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maximum of lead in solution, it breaks up into Zn and Ag_4Zn_5, and releases some of the lead, which sinks to the bottom. Under the same conditions, if held in solution by lead, the homogeneous alloy will be divided into the above components, the lighter zinc rising to the surface. The alloy Ag_4Zn_5 also dissolves lead, but to a smaller extent than a mixture of it with either $AgZn_5$ or free Ag would. It is, further, less soluble in lead than would be expected from the amount of zinc it contains. When exposed to the air it assumes a coppery hue. The low degree of solubility of Ag_4Zn_5 in lead which always retains some zinc explains perhaps more definitely what Roesing called the small affinity of silver for zinc-bearing lead.

Before zinc added to the base bullion can take up any quantity of silver, it combines with the gold and copper contained in the lead and saturates this, the amount taken up depending on the temperature of the lead (§ 11). By the use of zinc a market lead very low in copper is thus obtained, and by successive additions of zinc very small amounts of gold can be concentrated in a separate crust (the gold or copper crust) with some silver, and extracted at a profit. There is a difference of opinion* as to whether gold or copper combines first with the zinc. It would seem that it must be gold, as no desilverized lead is free from copper, but it never retains the least trace of gold.

In order to desilverize argentiferous lead with zinc successfully, it is necessary that the lead and zinc be reasonably pure. Tests made by Kirchhoff† on base bullion containing 4.5% foreign metals, such as copper, arsenic, antimony, bismuth, and zinc, showed that 2.87% zinc was required to desilverize the lead when the bullion had not been softened, while 1.75% was sufficient if softening had preceded the desilverization; the relative quantities of market lead produced were 43% and 72% of the bullion charged. The table on page 429 shows how the silver contents decreased with each addition of zinc.

With the crude lead the first five zinc additions served only to remove the impurities to such a degree that the desilverization could begin. That the first addition of zinc to the softened lead took up so little silver shows that the lead must have been very coppery.

* Percy, "Metallurgy of Lead," p. 174.

† *Metallurgical Review*, i., p. 234; Dingler, *Polytechnisches Journal*, ccxxviii., p. 265.

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Number of Additions.	Not Softened		Softened.	
	Ounces Silver per Ton.	Pounds of Zinc.	Ounces Silver per Ton.	Pounds of Zinc.
After drossing	85.60	.....	94.90	.....
1	85.50	250	85.60	150
2	85.30	250	47.60	150
3	88.80	150	16.10	150
4	88.50	100	1.70	150
5	88.00	100	0.18	100
6	48.20	100	.....	.....
7	8.20	100	.....	.....
8	0.80	70	.....	.....
9	0.15	30	.....	.....

Of the three metals, copper, arsenic, and antimony, that principally interfere with desilverization, antimony is the least objectionable, as lead with as much as 0.7% antimony, assaying 41 oz. silver to the ton, is desilverized in the Harz Mountains without being softened, 1.3% zinc being required, and 81.34% of market lead being produced.

Arsenic not only retards the desilverization greatly, but seems to prevent the zinc crust from separating satisfactorily from the lead. In skimming a kettle the usual beautiful, smooth, dark-blue surface is not seen, but it shows instead a rough, grayish-white surface; and even if skimming be continued until the lead solidifies, the surface will hardly change in appearance. Arsenic appears to follow the zinc more readily than antimony. In one instance* the relative proportions of arsenic and antimony in the base bullion were as 1 : 2.5, while in the retort bullion resulting from the distillation of the zinc crust they were as 1 : 0.25.

That copper combines with zinc before silver has already been mentioned.

Of the other two metals contained in Kirchhoff's bullion, zinc has a favorable effect; bismuth is indifferent, as it remains for the most part alloyed with the lead. Thus at one works† the bismuth in the refined lead rose from 0.0008 to 0.0233% during a period when bullion rich in bismuth was being refined.

Of the rarer metals tellurium‡ readily combines with zinc and enters the zinc crust. In desilverizing base bullion with 0.0025% tellurium, an addition of 0.25% zinc sufficed to carry 95% of it into the zinc crust.

* Heberlein, *Berg- und Hüttenmännische Zeitung*, 1895, p. 42.
† Junge, *Freiburger Jahrbuch*, 1895, p. 8.
‡ Heberlein, *Berg- und Hüttenmännische Zeitung*, 1895, p. 42.

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According to Balbach,* platinum and palladium, and according to Schnabel,† nickel and cobalt, also go into the crust.

The zinc used must be pure, if the desilverization is to proceed satisfactorily. The following analysis by Suppan,‡ of Missouri, Kansas and Illinois zinc show how pure is the spelter used:

	Glend'le, Mo.	Ch'e'kee, Kan.	Nevada, Mo.	Pittsb'g, Kan.	Peru, Ill.	Peru, Ill.	Peru, Ill.	Peru, Ill.	La Salle, Ill.
Pb	0.6581	0.6995	0.6725	0.4105	0.8723	0.3068	0.5875	0.3765	0.2513
Cd	0.0056	Trace.	0.0011	Trace.	Trace.	Trace.	Trace.	Trace.	0.0188
As	0.0353	Trace.	.....	Trace.	.....	.....	.....	.....	Trace.
Fe	0.0095	0.0815	0.0546	0.0523	0.0233	0.0383	0.0357	0.0390	0.0475
Ni	Trace.	.....	.....	Trace.	.....	.....	.....	.....	.....
S	Trace.	Trace.	Trace.	.....	0.0601	Trace.	Trace.	Trace.	.....
Bi	.....	.....	.....	Trace.	.....	.....	Trace.	.....	.....
Sb	.....	Trace.	.....	.....	.....	.....	Trace.	Trace.	.....

In experimenting with cheap zinc containing iron, obtained from galvanizing works, the writer found that the process was so retarded, and the quantity of impure zinc required so great, that no saving at all was effected by the use of the inferior material.

Jernegan§ records a similar experience, and Föhr‖ mentions that he required four times the usual amount of zinc, which was found to be due to its impurity. It contained 2.75% lead, 0.61% iron, 0.077% copper, and traces of tin, arsenic, antimony, cadmium, sulphur, and carbon; in good brands the iron appears only in the second decimal.

From the effect that foreign metals have on the result of the desilverization, it is clear that all argentiferous lead that contains them to any extent has to be softened. All or almost all American zinc-desilverizing works buy base bullion and ores in the open market to a greater or less extent. The bullion is therefore always liable to contain some arsenic or antimony, and every refinery softens the lead before it attempts to desilverize.

§ 102. Outline of Plant and Process.—The general plan of a desilverizing plant varies somewhat according to its location and the practice that prevails. All the arrangements, however, must be such as to require as little handling of lead and by-products as possible. In fact, the bulk of the lead, when charged into the

* Private communication, by A. W. Jenks, 1894.

† "Meta lhüttenkunde," vol. i., p. 530.

‡ "Bulletin Missouri Mining Club," i., No. 2, p. 51.

§ "Transactions of American Institute of Mining Engineers," ii., p. 288.

‖ *Berg- und Hüttenmännische Zeitung*, 1868, p. 28.

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softening furnace will in a modern plant never be handled again until it is ready for shipment. The result is that a section through a refinery will show the form of a terrace.

The general arrangement of a refining plant is given in plan and section in Figs. 338 and 339. On the highest level are the softening-furnaces, which receive the base bullion and prepare it for the kettles. The latter are a stage lower, and there the softened bullion is desilverized. The apparatus for liquating zinc crusts is also upon the same floor. In the drawing, each desilverizing-kettle has close to it only one liquating kettle, with a small kettle for liquated lead, showing that no distinction is made between gold and silver crusts. If the Howard press be used the liquating kettle or reverberatory furnace will not be required. The liquated crusts pass from this floor into an adjoining building, placed to the right or the left of the main building, the floor of which is on a level with the scale-floor. It contains two departments—the retort-room, where the crusts are distilled, and the cupelling-room, where the retort-bullion is turned into silver, or doré silver, bars. In the plant shown in the figure only doré bars will result. Following the desilverized lead, the refining-furnaces are reached on the next level, in which the desilverized lead is dezincified. Thence it passes into the merchant-kettles, and from these into the molds placed in the lead-pit. The market lead is loaded on trucks on the scale-floor that are of the same construction as the bullion-receiving trucks; they are run on scales, and the weighed lead is transferred into the cars on the loading-track. The plant for working the by-products will be placed on the side of the main building, opposite to where the zinc-crusts are treated. The manner of dealing with these varies greatly in different refineries, and will be discussed later on.

Roesing* draws a comparison between the arrangement of a refinery on a horizontal plane and that on an inclined plane, which, on the whole, is not favorable to the latter.

An illustrated description of the refinery at Tarnowitz, Silesia, which is built on a level plain has been published by Saeger.†

* *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xxxvi., p. 103; *Berg- und Hüttenmännische Zeitung*, 1888, p. 337.

† *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, 1893, xli., p. 268.

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According to the outline given, Parkes' process is best treated under the following heads:

Receiving Base Bullion.
Softening Base Bullion.
Desilverizing Softened Bullion.
Refining Desilverized Lead.
Molding Refined Lead.
Treatment of Zinc-Crusts.
Treatment of By-Products.
Table of Desilverization.
Conclusion.
Comparison between Pattinson's and Parkes' Processes.

§ 103. Receiving Base Bullion.—The base bullion arrives at the refinery in car-loads of from 14 to 20 tons. The track (receiving-track of Fig. 339) is laid so low that the bottom of the car is on a level with the "upper platform" of the works. Along the whole length of this platform, and parallel with the railroad, runs a narrow-gauge track of from 16 to 22-in. gauge, which bears a number of strongly-built low bullion-trucks. They may be built as follows. A frame consisting of two pieces of channel-bar iron, 3 ft. long, is fastened to the two axles of the wheels, and steadied by two iron bands running diagonally. The bullion is carried out from the car and loaded upon the truck standing before the car-door; when this is filled to a height convenient for lifting (about 3½ ft.), it is moved on, and another takes its place. The trucks are run on scales placed at one or two points in the "bullion and scale-shed," the bullion is weighed, and is then sampled from one truck directly upon another, which then moves straight to the softening-furnaces, or to places near them, and no more handling is required before the bullion is charged into the furnaces. The bullion produced in the smelting department of the refinery is loaded at the blast furnaces or reverberatory furnaces, on the same kind of truck, brought by an elevator to the "bullion and scale-shed," and then passes to the softening furnaces.

§ 104. Softening Base Bullion.—*Introductory Remarks.* The object of softening is to separate from the base bullion produced in the blast furnace, impurities, such as a copper, sulphur, tin, arsenic, antimony, etc., that would interfere with the desilverization. It comprises two processes, liquation and oxidation. By

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the former, metals and their compounds held in solution by the red-hot blast-furnace lead are separated out again from the readily fusible lead by melting it down slowly at a low temperature. By the latter, metals alloyed with the lead, and more easily oxidized than the lead, are removed by heating it to a bright-red heat with access of air, with the result that these metals are converted into oxides, and, combining with the lead oxide, are drawn off either as a powder or as a slag from the surface of the metallic lead.

When base bullion is melted down slowly in a softening furnace at a low heat, there rises to the surface a dark-colored, half-melted, pasty substance, the furnace-dross, consisting of a mixture of lead, copper, sulphur, arsenic, etc. The slower the melting down, the more effectual will be the separation of the copper from the lead. The following analyses show the purification effected in the lead by liquating and the composition of the dross when it has been freed as much as possible from adhering lead:

	Clausthal (*a*).		Lautenthal (*b*).		Freiberg (*c*).		Denver (*d*).	
	Before Drossing.	After Drossing.	Before Drossing	After Drossing.	Before Drossing.	Liquated Dross (5%)	Dross before Liquating.	Dross, after Liquating.
Pb....	98.92944	99.0289	98.96475	99.1883	96.667	62.40	53.0	50.0
Cu....	0.1862	0.1096	0.2838	0.0907	0.940	17.97	18.2	26.8
Cd....	Trace.	None.	Trace.	None.	.....	.....	.....	.....
Bi....	0.0048	0.0050	0.0082	0.0083	0.066	None.	.....	Au, 0.30 oz.
Ag....	0.1412	0.1420	0.1413	0.1440	0.544	0.17	.....	75.0 oz.
As....	0.0064	0.0053	0.0074	0.0082	0.449	2.32	.....	7.31
Sb....	0.7203	0.7066	0.5743	0.5554	0.820	0.98	.....	0.16
Sn....	None.	None.	None.	None.	0.210	0.04	.....	.....
Fe....	0.0064	0.0042	0.0089	0.0048	0.027	0.43	.....	.....
Zn....	0.0023	0.0017	0.0024	0.0015	0.022	0.07	.....	.....
Ni....	0.0023	0.0017	0.0068	0.0038	0.0055	1.09	.....	.....
Co....	0.00016	Trace.	0.00035	Trace.			.....	.....
S.....	.....	.....	.....	.....	0.200	4.00	2.0	3.6
O.....	.....	.....	.....	.....	.....	1.87	.....	.....
Slag, ash, hearth-material	.....	.....	.....	.....	.....	8.66	1.8	4.8

(*a*) Hampe, *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xviii., p. 203. (*b*) *Ibid.* (*c*) Schertel, *Berg- und Hüttenmännische Zeitung*, 1882, p. 293. (*d*) Iles, Private notes, August, 1896.

The analyses from Clausthal and Lautenthal demonstrate that the character of a comparatively pure lead is improved by melting down slowly and drossing. By comparing the two Freiberg analyses the degree will be seen to which the foreign matter of a very impure base bullion may be removed; viz., nearly all the sulphur, 96% copper, 99% nickel and cobalt, 25% arsenic, and only 5.8% antimony, and 1.54% silver. Bismuth remained entirely in the liquated lead, and all the tin excepting 0.9%.

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On melting down this dross in a crucible, Schertel* obtained a product consisting of lead, speise, and matte, in well-separated layers.

	Lead. Per cent.	Speise. Per cent.	Matte. Per cent.
Ag	0.34	.....	.....
Cu	1.79	37.60	47.70
Pb	96.50	25.68	32.80
Ni	0.08	8.60	0.25
As	0.75	27.00	1.15
S	.....	.....	17.72

The absence of iron in either speise or matte proves that these impurities do not result from finely-divided blast-furnace speise or matte being dissolved by the lead, as has been often thought. It tends to show that, being held in solution by the metal, the impurities unite on liquating to form a compound that is not fusible at the temperature at which the lead was melted down, and that the concentration of copper in dross is due probably to the presence of sulphur and arsenic and not to the separation of an alloy of lead and copper.

Heberlein† found that tellurium readily enters the dross and does not follow the arsenic and antimony into the skimmings as one might be led to expect from the similarity between it and these metals. He attributes this to its affinity for copper. In refining at Pertulosa, Italy, base bullion with 0.0025% tellurium the lead products contained the following amounts of tellurium:

Products.	Cu. Per cent.	As. Per cent.	Sb. Per cent.	Te. Per cent.	Ag. Per cent.	Au. Per cent.
Softening dross	34.29	.....	.....	0.0595	.....	.....
Softening skimmings	0.76	1.36	8.42	0.0185	.....	.....
Softened base bullion	0.23	0.051	0.12	0.0159	0.592	0.0012
Softened bullion after first zincking	.....	.....	.....	0.0019	.....	.....
Softened bullion after second zincking	.....	.....	.....	0.0008	.....	.....
Market lead (after third zincking refining)	.....	.....	.....	0.00065	.....	.....
Refining skimmings	.....	.....	.....	0 0010	.....	.....

If a sample, taken from the lead after drossing, be poured into a small mold, and allowed to cool slowly, a crystalline, bright, pewter-white spot will appear on the slightly depressed, dull,

* *Berg- und Hüttenmännische Zeitung*, 1882, p. 293.

† Ibid., 1895, p. 41; "The Mineral Industry," iv., p. 490.

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grayish-white surface, which in addition to the hardness of the lead is characteristic for the presence of arsenic and antimony.

In melting down base bullion that is very rich in copper, the amount of dross formed is sure to be large. As it carries with it precious-metal-bearing lead, it will reduce the direct yield from the charge below a practical limit. In order to prevent this, at some works galena is added to the softening-furnace charge for the purpose of forming a matte with the copper contained in the base bullion. The result is that all the gold remains in the metal bath and only a small percentage of silver is carried off by the matte. This modification in the method of removing the coppery dross has become important ever since the base bullion treated has assayed whole ounces of gold instead of tenths of ounces as was formerly the case.

If, after drossing in the usual way, the temperature be raised to a good red heat and the air permitted free access, the impurities contained in the lead will oxidize one after another: first tin, then arsenic and antimony. The surface will at first become quickly covered with dark-yellow skimmings, which vary from powdery to pasty, but are not fused on account of the tin; these are called tin-skimmings. They consist mainly of antimoniate and stannate of lead and antimoniate of tin, and are worked by themselves (§ 129). As soon as the tin-skimmings have been drawn from the surface of the lead, this begins to give off fumes of arsenic and antimony, and arseniate and antimoniate of lead begin to form. A sample of this fume gave Iles: As 2. 56%, Sb 32.73%, Pb 20.60%, Ag 7.0 oz., Au 0.05 oz. The arseniate of lead is lemon-yellow to light brown, the antimoniate dark brown to black; both are fused and drawn off together as antimony-skimmings when the furnace has been sufficiently cooled down for them to solidify. Toward the end of the operation the antimony in the skimmings will be replaced by lead until the black color has changed to the greenish-yellow of litharge.

Samples are then taken to see how far the softening has progressed. Before the antimony has been removed, a sample of the bullion taken in a ladle will "work," *i.e.*, small particles of melted black skimmings will float on the surface of the lead, with a rotary motion which resembles that of particles of grease on hot water. As the softening approaches the finishing point the globules become less in number and smaller in size, a thin coat-

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ing of yellow litharge forms more readily on the red-hot lead, and finally no more globules are seen and litharge forms quickly.

When a sample of the lead is poured into a mold, allowed to cool slowly, and skimmed with a flat piece of wood, it will, when it has solidified, have lost the characteristics of arsenic and antimony, and the surface of the bar will have assumed a rich indigo-blue color; the ease with which it can be scratched with the finger nail, and the luster on a freshly made incision, show the change that has taken place. It can also be tested by cupelling a small sample; if any incrustation is left on the cupel, the lead is not sufficiently softened.

§ 105. FURNACES.—The reverberatory is the furnace universally used in this country to soften base bullion. In some European works the liquation is carried on separately from the oxidizing smelting. The furnace is a real liquating furnace with an inclined hearth, from which the lead runs off into an outside kettle, whence it is ladled out ready for the softening furnace proper. The dross obtained is as free from lead as liquation can make it (see Freiberg analysis, *ante*). It is, however, more economical to make the entire softening process a continuous one, as it reduces the apparatus and the number of men necessary. Therefore, in American refineries, both liquating and oxidizing smelting are carried on in the same kind of reverberatory furnace, and the drosses which contain considerable lead are all liquated in a single liquation furnace, which is always kept running, and serves for the whole plant.*

In some works the bullion is melted down in the desilverizing kettle, drossed, heated to bright redness, and oxidized by introducing dry steam, which continually renews the surface of the lead, and thus hastens the elimination of arsenic and antimony.

This is the most expensive way. It consumes much fuel, is lengthy, forms a very large amount of oxidized product, and ruins the kettle. At the level, where the antimoniate of lead is in contact with the iron, it is eaten out. Even a kettle made especially thick at this place lasts only a short time.

The reverberatory furnaces used for softening are generally large enough to hold from 8 to 10% more bullion than the kettle into which they discharge their contents. About 15 years ago furnaces were constructed to hold from 15 to 20 tons of base

* For other methods of working, see "The Mineral Industry," iv., p. 482.

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bullion; their size has been gradually increased until most furnaces now hold from 33 to 35 tons of base bullion, furnishing 30 tons of softened bullion to the kettle. At some works, furnaces holding 50 and even 60 tons are in successful operation, but they form the exception.

The construction of the different furnaces is the same in many points. The hearth is elliptical or rectangular in plan; the length being to the width as $1\frac{1}{2}:1$, or often as 2:1. It is built of firebrick, inclosed by an iron pan to prevent the leakage of lead. In section it is dish-shaped; it is shallow, its depth varying from 11 to 16 in.; in exceptional cases it reaches 22 in. Its slope depends upon whether the tap-hole is located on the side or at the flue end of the furnace; it ranges from 2 to 5 in., the smaller figure always referring to side-tapping.

As regards detail, there is considerable variety in construction. The pan which is to hold the hearth used to be made of cast iron; it rested on transverse rails supported by brick walls running longitudinally. Thus the bottom of the hearth was effectually cooled by the air circulating beneath the pan. In order to relieve it from strains, the pan was allowed to stand free, the skewbacks supporting the roof being separate heavy castings, held in place by buckstays and tie-rods. Notwithstanding all these precautions, a cast-iron pan generally cracked after it had been in use for a little while. To-day there may still be found a few old furnaces with cast-iron pans; a new furnace, however, will always be built into a wrought-iron pan. The softening of lead that contains a few per cent. of antimony often takes considerable time and requires a pretty high temperature. Thus the brick suffers greatly from the corroding action of the antimoniate of lead and the litharge forming on the surface of the lead. The best firebrick soon begins to be eaten out if much hard lead comes to the furnace, and patching with a mixture of raw and burned fire-clay, or raw clay and coke, after every few charges, has to be resorted to in order to preserve the side walls. As the furnace has to be somewhat cool to make this repairing effective, much time is wasted. In many instances basic brick has been tried instead of acid firebrick, but its great expense has prevented its being more generally adopted. An improvement, using the ordinary firebrick, was made by introducing a 2-in. pipe between the two courses of brick which form the side-

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lining of the furnace, and allowing water to circulate through it. The inside course was eaten away by the litharge to a thickness of from 2 to 3 in. as quickly as before, but then the corrosion proceeded only very slowly, and the life of the side walls was thus greatly prolonged. With this encouragement the water-cooling has been carried to the extreme of inclosing the wrought-iron pan holding the hearth within another, leaving 3 or 4 in. space between the two, in which water circulates, thus cooling not only the sides but also the bottom of the furnace. A few of the softening furnaces in use to-day have these double pans with water circulating between them The outer pan is supported in the same manner as the former cast-iron pan; the inner pan rests also on rails which are laid in the same direction as the walls; stay-bolts connect the outer and inner pans. While without doubt this mode of cooling is very effective, there is unquestionably too much of it, considering the amount of fuel that is required to keep up the temperature necessary to soften the lead within the given time. It does check the corrosion of the hearth bottom by whatever litharge remains in the furnace after the base bullion or refined lead has been tapped, and is not floated up again by the next charge. Air-cooling alone has, however, always proved sufficient for the bottom; water cooling is necessary for the sides only.

The latest softening furnaces have water jackets only at the sides, and one of these, given in Figs. 340 to 346, is chosen to illustrate some of the details. The side elevation (Fig. 340) shows the fireplace *a* and the hearth inclosed by the water jacket *b* and the pan *c*. Between fire-box and hearth is left an air space *d* (Figs. 340, 341, 344). To counteract the bulging out of the pan and bending of the jacket, due to the expansion of the hearth, 3-in. I-beams are placed horizontally behind the jacket, and 7-in. I-beams behind the pan. They are not shown in the drawing. The discharge of the lead takes place at the flue end of the furnace through the spout *e*. The products of combustion pass off through the roof at *f* (Fig. 344) into the horizontal flue *g* (Fig. 342), which crosses the furnace, and then, passing down vertically close to it, leads into the main canal underground. The bullion is charged through two doors *h* on one side of the furnace (Fig. 341); the drossing and skimming take place on the opposite side, through three doors *i* (Figs.

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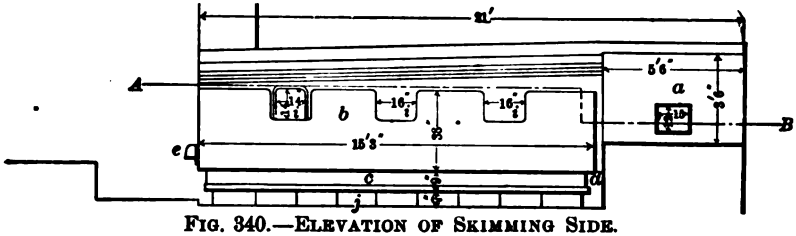


FIG. 340.—ELEVATION OF SKIMMING SIDE.

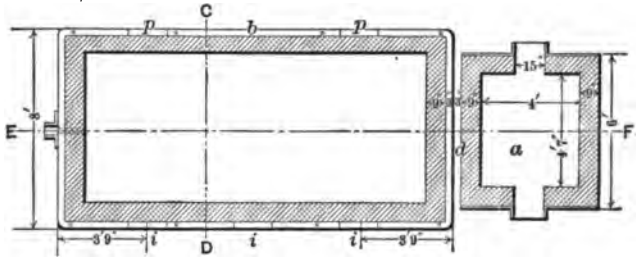


FIG. 341.—HORIZONTAL SECTION ON LINE A B.

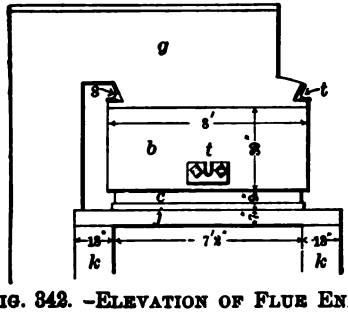


FIG. 342. -ELEVATION OF FLUE END.

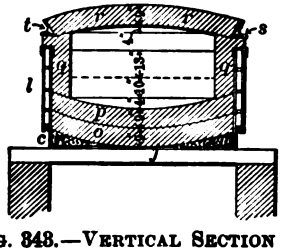


FIG. 343.—VERTICAL SECTION ON LINE C–D

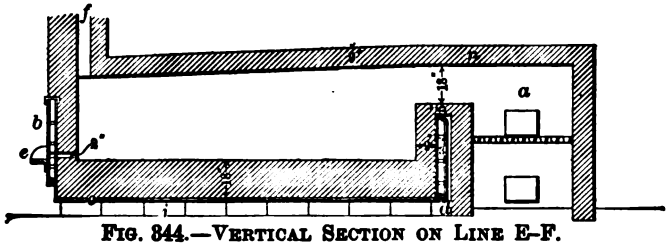


FIG. 344.—VERTICAL SECTION ON LINE E–F.

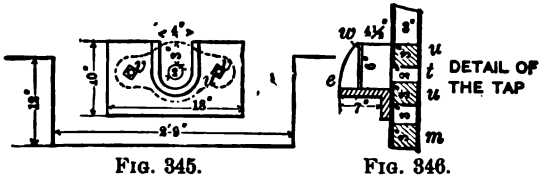


FIG. 345. FIG. 346.

FIGS. 340 TO 346.—SOFTENING FURNACE WITH WATER JACKETS FOR A 30-TON KETTLE.

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340 and 341). The charging-doors are on a level with the upper edge of the water jacket; the drossing and skimming-doors are let in 3 in. The pan of the hearth rests on 7-in. I-beams *j* placed transversely, and these rest on two brick walls *k* running longitudinally. The hearth (Fig. 343) is inclosed up to the level of the charging-doors, *i.e.*, to the depth of 42 in., by an iron pan made of $\frac{5}{8}$-in. boiler iron. The pan is surrounded by water jackets made also of $\frac{5}{8}$-in. boiler iron; their water space is 3 in. wide; the plates are kept apart by stay-bolts. The side jackets and the jacket at the flue end are 39 in. wide, while the jacket at the bridge end is 47 in. wide. The bottom of each jacket is formed by an inverted U-shaped piece of iron; the tops of the side jackets and of the jacket at the flue end are left open (Figs. 343 and 344), being covered only by loose pieces of bent sheet iron; the jacket in the bridge is closed (Fig. 344). The jackets have $1\frac{1}{4}$-in. water inlet and outlet pipes. To insure the complete filling of the bridge jacket, small pieces of pipe pass through the top. This further has hand-holes to clean out at intervals the mud that settles out from the cooling water. These details are not shown in the figures.

Another side jacket that is found at some furnaces has a somewhat different construction. It is open at the top and covers the entire side of the pan. Near the bottom it is slightly bent, and one row of rivets joins the bottom and side of the pan with the plate forming the jacket. Both kinds of jackets are in satisfactory use.

The manner followed in putting in the hearth (Fig. 343) varies somewhat at different works. Usually a layer of brasque is first carefully tamped in, and then so cut out that the course of brick, laid endwise upon it, shall bring it into the desired shape and give it the necessary inclination toward the tap-hole. Assuming this to be at the flue end of the furnace, as is most common, the thickness of the brasque there will rarely exceed 2 in., increasing to 5 in. at the center of the firebridge. In the figure very little brasque *n* is observed in the center; it increases to a thickness of 7 in. at the sides. It is made so thin because there are two courses of brick instead of one, as is usual. The lower course *o* is an inferior grade of firebrick, set dry, and grouted with a mixture of clay and cement so as to stop up the joints and prevent any percolation of lead. The upper layer *p* is made of

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the best firebrick available. In putting down the bottom the bricks have to be joined as tightly as possible. For this purpose they must first be carefully selected and fitted by rubbing them together until all roughness is removed. Each brick is dipped into water and then into a clay mortar having the consistency of very thin gruel, then put in place and driven with the hammer against the brick it is to face. This makes the joint as close as possible and prevents the passage of lead. The sides of the furnace are built with the same care as the bottom. Commonly they rest on the curved working bottom to prevent this from rising. The roof is supported on either side by a 60-lb. rail *s* and a 4-by-1-in. plate *t*., which is held in place by wedges between it and the buckstays. The furnace is bound in the usual way with buckstays (old rails) and tie-rods. The manner of tapping the furnace deserves special mention. In Fig. 342 the water jacket is seen to inclose the tapping-opening. Details are shown in Figs. 345 and 346. The 2-in. tap-hole *t* is a conical opening in the cast iron plate *u*, which fills the open space between the outer and inner plates of the jacket. Two bolts *v* pass through the jacket and plate, and then through the flanges of the spout placed on the outside, where they are tightened with nuts. Commonly the tap-hole is closed by simply ramming a clay plug into it. To make the breaking away of the clay impossible, the tap-hole in the figure is closed by an iron plug coated with clay. It is held in place by an iron wedge driven vertically between it and a horizontal piece of flat iron, held in place by the vertical ribs *w* on either side of the spout.

Finally, as to the firing of the furnace. The fuel commonly used is bituminous coal; at many works this has been replaced by crude oil. With a good grade of bituminous coal, natural draft is sufficient to soften the lead in the required time. Many works use slack coal, and then blast under the grate becomes necessary. At some works a series of small blast pipes is introduced through the roof, above the firebridge, with very good effect. The writer recalls one instance where, with the use of blast both under the grate and in the roof, and an impure bituminous coal, Colorado base bullion of ordinary hardness was softened in 50-ton charges in six hours. This is probably the best work on record. The good effect of cold blast in the roof would suggest the admission of hot air, as is done in Lake

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Superior copper refining furnaces,* where a flue ascending in the side wall of the fireplace passes through the roof, and enters the furnace just above the bridge, where it delivers the heated air to effect complete combustion. If necessary, additional heated air might enter the furnace through openings in the firebridge.

In the use of oil as fuel, air under a pressure of 8 or 9 oz. has replaced most of the different forms of Körting inspirators, as injection with compressed air proved a saving of oil when compared with steam injection, 100 gal. with air being equal to 145 gal. with steam. The Reed burner of the Standard Oil Company is the one in common use.

§ 106. Mode of Conducting the Process. — The mode of operating the softening furnace is about as follows: The bullion is charged through the two charging-doors by means of a long-handled paddle, and melted down slowly.

The paddle is a rectangular iron bar about 8 ft. long, made of 1½-in. iron, one end being flattened out for a distance of 2 ft. 6 in. to the width of 3 in., to receive a bar of bullion, while the other is rounded off and bent to a ring. A lug is often cast on either side of the door-frame to support a roller. It serves as bearing for the paddle instead of the door-frame, and thus facilitates the manipulation.

In some works the bullion is all charged at once; in others, where the charging, softening, desilverizing, refining, and molding are given in contract to a crew of four men, the bullion is charged at intervals when the kettle and refining furnace do not require their attention. The softening of the base bullion, as well as the refining of the desilverized lead, is regulated by the desilverization which takes the longest time (from 16 to 18 hours for a 30-ton kettle). When a kettle of bullion has been desilverized, the refining furnace must be empty to receive the desilverized lead (average time required 8 hours), and the softening furnace ready to furnish new softened bullion to the kettle (average time required 8 hours). Upon this general scheme the whole work of the refinery must be based to be properly continuous.

The bullion is melted an hour or more after it has been charged. It is stirred to detach some of the lead held in suspension by the dross; sometimes fine coal is spread over it and stirred in. This is very effective when the bullion is pure, so

* Egleston, "Transactions of American Institute of Mining Engineers," ix., p. 690, plate ii.

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that little dross rises to the surface. With impure bullion considerable fuel is required to have any effect, and there is danger of the temperature becoming too high and the lead taking up again some of the impurities that had separated out. The dross is removed by a rabble or a rectangular skimmer.

The head of the rabble is made of $\frac{3}{8}$-in. iron, and is 3 by 12 in., the handle of $\frac{7}{8}$-in. iron, and 10 ft. long. The handle of the skimmer is of the same length and thickness as that of the paddle; the perforated part is made of $\frac{1}{4}$-in. iron, and is 10 by 12 in., the perforations being $\frac{3}{8}$ in. in diameter.

With either tool about the same amount of lead is withdrawn with the dross from the furnace, so there is little choice between them; some prefer one, some the other. The handle often rests in a hook suspended by a chain from the roof, thus facilitating the work. The operator removes from one side the dross, which his helper on the other side collects with a rabble, pushing it toward the door or upon the skimmer. The dross, while being removed from the furnace, is collected either in a slightly conical cast-iron mold running on wheels, *e.g.*, 2 by 3 ft. at the base and 14 in. deep, made of $\frac{5}{8}$-in. iron, or preferably in an iron two-wheel barrow with perforated bottom, in order that some of the lead carried out with the dross may run off on the cast-iron plate in front of the skimming-doors and be returned to the furnace. The use of galena with bullion rich in copper has already been referred to above.

The dross drawn off is weighed, and a sample is taken from the lead remaining in the furnace to be assayed. The weight and assay-value of the bullion charged being known, the weight of the dross and the assay of the residual hard lead give the data necessary to calculate the total silver contained in the dross, and with it its assay. The amount of dross formed is about 4% of the bullion charged, and assays about 80% of lead. It is freed from some of its lead in a liquating furnace, and will then have a composition similar to that of the Freiberg analysis (see *ante*).

The tin-skimmings that form on raising the temperature after drossing are removed in the same way as the dross.

With the antimony-skimmings it is customary to cool the furnace by throwing open the doors, in order that the antimoniate of lead floating on the surface may harden, and be then taken off

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in the form of a thin crust. If the bullion is very hard, skimming once will not be sufficient to soften it. The furnace is therefore heated up again, and as soon as the surface of the lead is well covered, the cooling and skimming are repeated; ordinarily two operations are sufficient, but sometimes three are necessary. To hasten the cooling of the furnace, slacked lime is sometimes spread over the metal bath. Some refiners add lime to the furnace after drossing, with the idea that an antimoniate of lime is formed, and thus less lead is oxidized during the softening. This effect of lime still remains to be proved. Any addition of lime to the furnace has the great disadvantage of interfering with the subsequent liquation of the antimony skimmings, and is therefore better dispensed with altogether.

If the bullion is very hard, the addition of litharge from the cupelling furnace greatly shortens the time required for softening.

Another method of hastening the softening is the introduction of dry steam to stir the lead, thus continually exposing a fresh surface to the oxidizing action of the air. This is done by introducing through each of the charging-doors a 1-in. pipe, to the end of which are screwed, by means of a T, two pipes having a number of perforations on either side and closed at the ends. The main pipe is bent so that when it is introduced into the furnace and held in place by the closed furnace-door, which has been weighted, the two pipes at its ends will be pressed down into the lead, and run parallel to the sides of the furnace. While the introduction of steam does shorten the time required for softening, it has the disadvantage that it forms a large amount of skimmings and that the swash of the lead oxide and antimoniate strongly corrodes the sides of the furnace. It is, therefore, to be used only in extreme cases.

A third method to be mentioned is the one in use at Freiberg, where bullion rich in tin, arsenic, and antimony is softened. Blast is introduced on either side of the firebridge, and the skimmings are removed at the flue end of the furnace as fast as they form. The tool used is a long iron hook, to which is fastened a triangular piece of wood, say 8 in. long. With it the skimmings are drawn out of the furnace in a thin stream. In order to facilitate the work, and to enable the workmen to pass gently over the surface, and thus remove only skimmings, but

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no lead, the handle is supported by a hook suspended from the roof.

After the last skimmings have been removed, the doors are thrown open to cool the lead before it is tapped into the kettle. The skimmings are weighed and a sample of the softened bullion in the furnace is taken for assay; thus the silver contained in the skimmings can be calculated. The amount of skimmings found is about 5% of the weight of the bullion charged. The fuel consumed for the entire softening is about 156 lb. of soft coal per ton of bullion charged, or 8.6736 gal. reduced oil, air being used as atomizer.

Samples of fluedust from softening furnaces showed the following values:

Pb.	Zn.	S.	SO_3.	As.	Sb.	Ag, Oz.	Au, Oz.	Authority.
52.2	2.8	25.1	12.2	Trace.	0.56	76.6	0.16	M. W. Iles.
33.8	4.4	.....	.....	.....	.....	30.0	0.16	M. W. Iles.
15.9	.....	.....	.....	.....	.....	9.1	0.16	F. B. F. Rhodes.

A new furnace bottom absorbs a considerable amount of base bullion for which it is difficult to give any figure. One peculiarity still needs a satisfactory explanation, namely, that a larger proportion of gold collects than of silver, considering the average composition of the bullion treated.

§ 107. Desilverizing Softened Bullion. *Introductory Remarks.*—From the softening furnace the lead, when sufficiently cool, is tapped into the desilverizing kettle, which has been whitewashed with lime water and heated to the point where a splinter of dry wood thrown on the bottom will ignite readily. The whitewashing facilitates the removal of silver crusts which adhere to the sides when the kettle is cooling. If the lead were tapped into a cold kettle, this would be liable to crack on the bottom, and the time for bringing the lead up to the required temperature would be unnecessarily prolonged. The lead runs into a trough of cast iron, $\frac{3}{8}$ in. thick, placed beneath the discharge-spout of the furnace. In order to decrease the amount of dross, the lead runs from the trough into a cast-iron pipe placed upright in the kettle.

The kettle-dross formed amounts to about 1% of the bullion charged. It is skimmed off and added to the next charge in the

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softening furnace after the furnace-dross has been taken off. The kettle is now ready for the addition of zinc.

The quantity of zinc necessary varies according to the purity of the lead, and increases on the whole with the amount of silver present. Roswag's* new formula is

$$Z = 10.39 + 0.035\ T,$$

where Z = kilogrammes zinc to be added to one metric ton of lead, and T = grammes silver in 100 kilogrammes lead.

This corresponds to

$$Z' = 23.32 + 0.223\ T',$$

where Z = pounds zinc to be added to 2,000 lb. of lead and T = ounces silver per ton.

Roswag also formulated the quantities for the zinc Illing† found necessary to desilverize lead of varying tenor in silver. They are:

Z'' kilogrammes $= 11.66 + 0.0325\ T''$
(grammes silver in 100 kilogrammes lead)

for every metric ton of base bullion; which corresponds to

Z''' pounds $= 20.78 + 0.24\ T'''$ (ounces silver per ton)

for every 2,000 lb. of base bullion.

These general formulæ probably give an approximate idea of the total amount required to desilverize base bullion running low in silver, say 30 oz. For rich bullion the figures are too high. How the amount of zinc to be added increases with the silver contents, irrespective of the zinc recovered later by distillation, is shown in the following figures given by Plattner:‡

Assay of Base Bullion. Ounces per Ton.	Per Cent. of Zinc Added.
28.09	1.84
111.56	1.84
148.16	1.96
245.00	2.45

In practice it has so far not been found possible to desilverize a rich base bullion with a single addition of zinc. The richer the bullion the less difference is there between the assay-values

* "La désargentation de plomb," Paris, 1884, p. 241.

† *Zeitschrift für Berg-, Hütten und Salinen-Wesen in Preussen*, xvi., p. 61.

‡ *Berg- und Hüttenmännische Zeitung*. 1889, p. 117.

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of the zinc-silver crust and the residual lead. From three to four zincings are therefore necessary. Low-grade bullion can be and is desilverized by two zincings. The aim in desilverizing must be to concentrate as much silver as possible into one zinc-crust, so as to utilize all the power of the zinc. This is best done by first adding sufficient zinc to remove the gold and the copper (as gold crust) with as little silver as possible, and to saturate the lead; then the bulk of the zinc is added, which takes up enough silver to form the rich zinc-silver-lead alloy (the first silver crust). One, or occasionally two, subsequent additions of zinc will completely desilverize the lead, but the zinc contained in these crusts will, under suitable conditions, combine with more silver; hence the second and third silver crusts are added as fresh zinc for the first desilverization, and two-thirds of the zinc contained in them is available as new zinc. The crust obtained from a kettle assaying 60 oz. or less per ton always goes back to the kettles. The second silver crust ranges ordinarily at from 18 to 30 oz., the third silver crust from 0.3 to 0.5 oz. per ton, while the first never ought to run lower than 2,000 oz. The following analyses show the composition of some silver crusts, obtained from low-grade bullion that has been drossed only and still retains the antimony, and of three liquated gold crusts from softened bullion rich in silver and gold:

	Altenau. (a)	Lautenthal. (b)	Mechernich. (c)			Friedrichshütte. (d)		Freiberg. (e)	Denver (f)	Newark (g)
	1880.	1880.	1875. (c)	1884. (d)	Average. 1884 (d).	1869.	1884.	——	1896.	1895.
Pb.....	75.675	77.880	48.80	49.70	45—48	34.66	81.2	52,300	67.00	31.38
Zn.....	11.78	12.11	39.00	34.00	35—45	20.19	12.15—10.15	39.700	18.20	22.05
Cu.....	1.12	0.82	5.33	6.00	2—3	.....	.....	2.680	4.40	45.82
Ag	1.855	2.420	1.22	1.75	1.5—1.7	1.21	1.075—1.202	4.015	2.96	0.380
Au.....	.....	.....	.....	.....	.....	.....	.....	0.0153	0.0108	0.2774
As.....	Trace.	Trace.	.....	.....	.....	.....	.....	.....	.....	.....
Sb.....	.....	.....	0.36	.....	.....	0.83	.....	.....	.....	.....
Cd.....	Trace.	Trace.	.....	.....	.....	.....	.....	.....	.....	.....
Ni	Trace.	Trace.	.....	.....	.....	.....	.....	.....	.....	.....
Fe.....	.....	.....	1.28	.....	.....	0.96	.....	.....	.....	.....
PbO.....	4.75	4.00	.....	.....	.....	.....	.....	.....	.....	.....
ZnO.....	0.60	0.44	.....	.....	.....	.....	.....	.....	.....	.....
Bi_2O_3...	1.72	0.37	.....	.....	.....	.....	.....	.....	.....	.....
H_2O_3....	0.68	0.96	.....	.....	.....	.....	.....	.....	.....	.....
Fe_2O_3 ..	1.87	1.04	.....	.....	.....	.....	.....	.....	.....	.....

(a) *Zeitschrift für Berg, Hütten- und Salinen-Wesen in Preussen*, xxviii., p. 262. (b) *Ibid.* (c) *Berg-, und Hüttenmännische Zeitung*, 1875, p. 129; *Engineering and Mining Journal*, March 17, 1877. (d) *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*. xxxiv., p. 92. (e) Schnabel, *Metallhüttenkunde*, i., p. 547. (f) Iles, private notes, 1896. (g) Jenks, private notes, 1895.

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Of the gold crusts the one analyzed by Jenks (*g*) shows an abnormal composition, as the retort bullion assays ordinarily 2,000 oz. silver and 5 or 6 oz. gold per ton. This crust resulted from bullion containing as impurities Cu 0.37%, As 0.02%, and Sb 0.15 %. It looked like small sponges of brass floating in the normal gold crust. They formed to the amount of several hundred pounds and seem to be a true alloy.

All the German silver crusts given run much lower in silver than any from American refineries. Those from Altenau and Lautenthal retain more lead than any of the others, as they are to be melted down again in a kettle and decomposed by steam, which could not be satisfactorily done if the liquation had been carried any further.

It is usually stated that the total zinc required is added in three separate portions: first two-thirds, then one-fourth, and finally the remaining one-twelfth. While this corresponds approximately to actual practice, it cannot be implicitly followed, as the amount of the successive additions must depend on the assay of the bullion. The rate at which the silver in the lead decreases with the additions of zinc is influenced by a number of circumstances that cannot be determined in advance. Each refinery has its own table to show the amount of zinc that shall be added if the bullion assays a certain number of ounces of gold and silver to the ton. Some of these tables are very complicated. The following is one of the simplest. By it, the gold and silver from a 30-ton kettle are extracted separately with three zinc-additions.

TABLE OF ZINC-ADDITIONS FOR GOLD.

Up to 0.10 oz. gold per ton, 250 lb. zinc.	0.50—0.70 oz. gold per ton, 400 lb. zinc.
0.10—0.30 oz. gold per ton, 300 lb. zinc.	0.70—0.90 oz. gold per ton, 450 lb. zinc.
0.30—0.50 oz. gold per ton, 350 lb. zinc.	etc. etc.

It is to be noted that gold and copper are extracted from the lead without saturating this with zinc. For instance, 30 tons of lead, taking up 0.6% of zinc, require 360 lb. of zinc, while 0.30 oz. gold per ton are extracted by an addition of 300 lb.

When the gold crust has been skimmed off the kettle, 500 lb. of zinc are added for bullion assaying from 150 to 250 oz. silver. With bullion running as high as 300 or 400 oz., 550 lb. are given. After removing the first silver crust the kettle will assay from 10 to 50 oz. silver per ton, generally from 30 to 40 oz.; in excep-

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tional cases it may run as high as 70 oz. The second and final addition of silver-zinc, varying from 400 to 600 lb., will reduce the silver contents of the kettle down to a trace, even if it has been as high as 70 oz. Bullion running low in gold and silver, say from 0.05 to 0.10 oz. gold and from 50 to 125 oz. silver to the ton, receives one gold-zinc and one silver-zinc, the resulting lead running less than 0.2 oz. silver to the ton and generally a trace.

The following table for a 30-ton kettle by Eurich forms the basis of most of the zinc tables:

ZINC TABLE FOR A THIRTY-TON KETTLE.

Second Addition to Bring the Silver Contents to 40 Oz.		Third Addition to bring the Silver Contents to 1 Oz.		Fourth Addition to Bring the Silver Contents to 0.1 Oz.	
Oz. Silver per Ton.	Pounds Zinc.	Oz. Silver per Ton.	Pounds Zinc.	Oz. Silver per Ton.	Pounds Zinc.
40	15	5	225	0.2	60
50	50	7	265	0.3	75
60	100	10	320	0.4	90
70	160	15	400	0.5	105
80	200	20	450	0.6	120
90	245	22	470	0.7	135
100	285	24	485	0.8	150
110	315	26	500	0.9	165
120	345	28	512	1.0	180
130	365	30	530	1.5	225
140	390	32	540	2.0	330
150	415	34	555	2.5	390
160	440	36	570	3.0	450
170	460	38	585	3.5	510
180	475	40	600	4.0	562
190	495	42	615	.....	.....
200	515	46	630	.....	.....
.....	.....	48	640	.....	.....
.....	.....	50	655	.....	.....
.....	.....	52	670	.....	.....
.....	.....	54	680	.....	.....
.....	.....	56	695	.....	.....
.....	.....	58	710	.....	.....
.....	.....	60	784	.....	.....
.....	.....	62	747	.....	.....

It starts with the second addition of zinc presupposing that by the first zincing the lead has been saturated with zinc and all the copper (and gold) removed. Assuming that 500 lb. of zinc and the last crust of a previous charge have accomplished this, and the kettle assay has given 146 oz. silver per ton: The table shows in the first column that 140 oz. require 390 lb. of zinc; 400 lb. are given; they reduce the silver contents to 28 oz. The second column calls for 512 lb. to bring the silver contents down to under 4 oz. per ton. If the kettle assay gave 1 oz. silver per ton, there would be required, according to the third column, a

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fourth addition, viz., 180 lb. to clean the kettle. By giving in the third zincing 612 lb. instead of 512 lb., *i.e.*, an additional 100 lb., the silver contents will be brought to below 0.1 oz. per ton, and thus the fourth zincing avoided.

Lately Edelmann and Rössler* have carried on an interesting series of experiments with the object of concentrating the silver in a richer alloy than usual. The ordinary silver crust, they say, consists of a small quantity of zinc-silver alloy distributed through zinc-bearing lead which is partly oxidized. As the oxidized part obstructs a satisfactory separation of the zinc-bearing lead from the zinc-silver alloy, the improvements must consist in preventing the formation of oxides; then the direct output of desilverized lead would be increased, an alloy rich in silver produced, and the consumption of zinc reduced. The addition of 0.5% aluminum to the zinc, in some of their experiments, had a favorable effect in preventing oxidation, and thus assisted in obtaining crusts very rich in silver.

It was, however, necessary that the lead to be desilverized should be free from copper and arsenic; 0.1% copper and 0.05% arsenic neutralized the effect of the aluminum, 0.03% antimony was not harmful, but with 1% antimony the desilverization was unsuccessful. In working on a large scale at Hoboken, near Antwerp, and at Lautenthal, in the Harz Mountains, the process proved unsatisfactory, as beside the gold crust, which had to be heavy to insure the removal of the copper to below 0.1%, two silver crusts were obtained—a small one, pretty free from oxide and rich in silver, and a large one resembling the ordinary silver crust. On liquating the two together, about equal parts of rich unoxidized alloy with 20% silver and 4% lead, to be refined electrolytically or dissolved in sulphuric acid, and of relatively low-grade partly-oxidized alloy, to be retorted in the usual way, were obtained. While the desilverization worked satisfactorily, difficulties were encountered in the working of the crusts. The electrolytic refining of the rich crust was too expensive and treatment with sulphuric acid was very slow, as the granulated alloy did not dissolve readily and it was next to impossible to get enough copper into solution to obtain a silver bullion of sufficient

* *Berg und Hüttenmännische Zeitung*, 1890, pp. 245, 429; 1891, p. 123; *Engineering and Mining Journal*, Nov. 15, 1890; April 4, May 16, 1891; Sept. 2, Oct. 21, Dec. 2, 1893; Schnabel, *Metallhüttenkunde*, i., p. 573.

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high grade. Further there is only a small market for zinc vitriol. According to Schnabel* the process has been given up at both places.

Hasse† experimented upon desilverizing base bullion by means of an alloy of zinc and magnesium which proved successful on a small scale as far as the desilverization was concerned.

Other suggestions regarding improvements in desilverization have been made by Rösing,‡ Honold,§ and Schlapp.‖

The only real practical advance in enriching the zinc crust is the introduction of the Howard stirrer and press (see later).

§ 108. DESILVERIZING KETTLES.—The kettles used for desilverization are spherical in form. They are from 3 ft. to 3 ft. 4 in. deep, and their diameter varies according to the required capacity. Most kettles used to-day hold 30 tons; the dimensions of such a kettle are shown in Figs. 347 to 376. A circular kettle of a greater capacity would be difficult to work by hand, although 60-ton circular kettles are in successful operation using the Howard stirrer. In the few instances where hand-stirred kettles holding over 30 tons are used, they are made oblong and have rounded or elliptical ends. An oblong kettle, holding 55 tons of lead when actually filled, and 45 tons when filled to 5 in. from the top, is 12 ft. long, 7 ft. wide, and 3 ft. 3 in. deep. An elliptical kettle working 60-ton charges is 10 ft. 10 in. long, 8 ft. 9 in. wide, 3 ft. deep, and 2 in. thick. The thickness may be less, which will make the kettle cool more quickly.

The kettles are all of cast iron. The iron used should be dense and strong, but not hard. Lake Superior No. 2 iron, which is neutral, strong, and tough, mixed with car-wheel iron, giving the necessary density, furnishes a material filling every requirement. Kettles are best cast bottom down, as it gives greater density. The other way is easier for the foundry. In any case they should be cast in brick-work if a smooth surface is to be obtained, which is necessary for the scraping of the kettle; any blisters, beside weakening the kettle, are liable to assist the corroding effect of the zinc, and to retain particles of zinc crust

* *Berg- und Hüttenmännische Zeitung*, 1897, pp. 39 and 40.

† *Ibid.*, 1895, p. 483; "The Mineral Industry," iv., p. 488.

‡ *Chemiker Zeitung*, 1889, p. 1059.

§ *Berg- und Hüttenmännische Zeitung*, 1890, p. 187; 1891, p. 342; 1893, pp. 21, 51; *Stahl und Eisen*, 1891, p. 152.

‖ U. S. Patent, No. 380,524, April 3, 1888.

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which may enrich already desilverized lead. Kettles have often been made 2½ in. thick at the bottom, tapering to 1½ or 2 in. at the rim. At present they are made of uniform thickness throughout, and rarely over 1½ in. in thickness. Such a kettle lasts from one to one and a half years, being in continual use. Steel kettles are not found in American desilverizing works. They have come into use in European works, where the lead is refined by means of steam in the same kettle in which it was desilverized. A kettle is usually suspended by its rim, which rests on a circular cast-iron ring covering the top and sides of a brick wall. Figs. 367, 371, 372 show three supporting rings; see Fig. 367 for the desilverizing kettle, Fig. 371 for the liquating kettle, and Fig. 372 for the liquated-lead kettle. The casting (Fig. 367) consists of four separate pieces, shown in section by Fig. 368. They are fastened together by bolts passing through flanges, as seen in Fig. 369. The casting rests on the working-platform of the kettle, as drawn in the front elevation (Fig. 347). It is made 9¾ in. wide, and incloses the 9-in. wall, which rises 18 in. above the main brick-work. With many kettles the support of the rim consists of a circular iron ring, covering only the top of the brick-work which must be thicker than 9 in., if it is not to give way to the pressure of the weight of the kettle filled with lead. The side wall thus often reaches a thickness of 18 in., the iron support-ring either entirely covering it or leaving 2 or 3 in. exposed. Müller* gives it as his experience that a kettle lasts longer if suspended from a rib cast on the kettle at half its depth. This rib would then form the partition wall between the fireplace and the encircling flue, which simplifies the construction of the brick-work. The place from which the kettles are fired is seen in the front elevation (Fig. 347). The horizontal section (Fig. 350) shows the plan of the brick-work with the ash-pits of the three kettles.

Figs. 348, 349, 351 give more detail of the brick-work, and show the road of the products of combustion from the grate to the flue leading to the chimney. In the desilverizing kettle the flame goes from the fireplace *d* (Figs. 348, 351), first back and upward; it then passes around the kettle to the right, in a circular flue (as indicated by the arrows), and leaves this at *e*, entering a vertical flue leading to the main chimney. In the

* *Berg- und Hüttenmännische Zeitung*, 1889, p. 218.

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liquating kettle the products of combustion go from the fireplace *f* (Figs. 349, 351), after passing under the kettle, straight into the flue *g*. The gases from beneath the liquated-lead kettle *h* (Fig. 351) go to the left, and join those of the liquating kettle.

Desilverizing kettles were formerly emptied by a discharge-pipe cast in the bottom of the kettle and running out through the brick-work. It was closed either by a slide-valve on the outside (similar to the lead discharge of the Luce-Rozan crystallizer), or by a clamp and thumbscrew on the inside. At present the Steitz siphon is in common use, and is preferable to any other means for emptying a kettle. A common form of it is shown in Fig. 388. It consists of a piece of gas-pipe *a* from 2 to 2½ in. in diameter, bent so as to reach from the rim of the kettle to the bottom. Here it has an elbow *b* screwed to it to prevent the lead column from breaking. To the other is attached, also by an elbow *c*, the vertical section-arm *d*, having a cast-iron stop-cock *e* near the lower end. The siphon discharges the lead into a cast-iron trough of ⅜-in. iron, which carries it into the refining furnace. In order to shorten the arm reaching into the lead and to increase the heating surface, the bottom of the kettle is sometimes made convex in the center.

§ 109. Liquating Apparatus. — In connection with the desilverizing kettle must be discussed the apparatus required for liquating the zinc crusts. Three sorts are in use. The first is shown in Figs. 374 to 376. It consists of a shallow kettle placed on the same level with the desilverizing kettle and close to it. The bottom is convex in the center, in order that the discharging spout may be shorter than would be possible if the kettle had the usual spherical form. Formerly a perforated cast-iron disk (an old skimmer) was placed inside the kettle, over the opening into the spout, to prevent particles of crust from passing off with the liquid lead; this has been given up at some works, as the perforations easily become clogged, and when open do not prevent fine particles from being carried off by the liquated lead. This runs from the spout into a small spherical kettle, whence it is bailed out after it has been skimmed.

The drawings given show for every desilverizing kettle one liquating kettle with its liquated-lead kettle. This presupposes that no distinction is made between gold crusts and silver crusts, with the result that all the silver produced contains gold, and has

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to be parted. Where the crusts are kept separate, the desilverizing kettle will have a liquating kettle on either side, one for the gold crust, the other for the silver crust, the liquated crusts as well as the liquated lead from the two small spherical kettles being kept separate. Only the silver resulting from the gold crust will then have to be parted. With a plant where the two crusts are kept separate, the distance between the centers of the two desilverizing kettles of 32 ft., as shown in the general plan (Fig. 338), will be too small; it will have to be enlarged sufficiently to leave a passageway between the two small liquated-lead kettles.

The second apparatus is a reverberatory furnace placed on the floor of the desilverizing kettles. It may be built as follows: The hearth is a cast-iron plate, 10 by 5 ft., slightly trough-shaped, and having a rim 4 in. high along the sides. To the lower end a spout is attached, which discharges the liquated lead outside of the furnace into a small liquated-lead kettle of the same form as the one described, the rim of which is on a level with the working floor. The plate has an inclination of 3 in. It lies on a bed of brasque, which is tamped into a wrought-iron pan supported by transverse rails resting on two longitudinal walls. The furnace has two working-doors on one side. The writer prefers the reverberatory furnace to the liquating kettle, as he obtained from it in a shorter time than from the kettle a dryer crust, and thus a richer bullion, without driving some of the silver from the crust back into the liquated lead, as often happens with the kettle. The reason for the better result in the reverberatory furnace is probably that the lead can be gradually eliminated at a slowly increasing temperature in a reducing atmosphere, which prevents the oxidation that always takes place in a liquating kettle, even if it be covered with a sheet-iron plate, on raising the temperature to the required degree. Supposing, however, the liquation to be equally good with the two apparatus, there remains this advantage for the reverberatory, that the liquation of the silver crusts of several systems can be performed continuously in one furnace, which, being separated from the desilverizing kettles, does not disturb the work there, and which collects all the rich crusts into one place, whence they are delivered through a chute into the bins of the retort-room. With a small plant a liquating reverberatory furnace of any rea-

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sonable dimensions would not have sufficient work to do to pay for the extra labor. The liquating kettle must be used, even if it obstructs to some extent the work in the desilverizing kettle. The preference for the reverberatory furnace is not general, as several important refining works adhere to the liquating kettle.

The third and latest device is the Howard alloy press,* shown in Figs. 377 and 378, which has already in many works replaced the liquating kettle or the reverberatory furnace. The press consists of a cast-iron cylinder *a*, 22 to 26 in. in diameter; with a perforated hinged bottom *b*, and a screw-press *c*, operated by means of worm-gear *d*. The whole is inclosed in the frame *e*, and suspended from a traveler *f*, so that it can serve a number of kettles. It is generally raised and lowered into the kettle *g* by means of a differential pulley *h*. Sometimes this is replaced by compressed air or an electric motor. The method of working is as follows: The press is brought on the overhead track over the kettle, lowered into the lead and allowed to remain there until it has assumed the temperature of the lead. It is then slightly raised and the crust pushed or skimmed into the cylinder. When filled it is raised, the crust stirred with an iron rod to assist the running off of the lead, and pressure applied. All the unalloyed lead flows back into the kettle whence it came. The press is now pulled to one side and the bottom dropped, when the cake of compressed alloy will fall to the floor, where it is easily broken up into pieces suited for the retorts (see § 111). The operation is repeated until the kettle has been skimmed. The advantages of the press are, that it does away with liquating kettles and reverberatory furnaces as well as all extraneous heat, thus giving more room around the kettles and simplifying and cheapening the operation (one press serving a number of kettles), that not being oxidized the crust gives up its lead readily, and that the liquated lead flows directly into the desilverizing kettle where it belongs.

The practice of liquating zinc crusts in a spherical kettle without a lead-discharge and removing the liquated crust floating on the liquid lead with a skimmer is antiquated, if the crust is to be distilled, and justly so, as it is impossible to obtain in this way a dry crust that is satisfactory. If the temperature of the lead be raised sufficiently high to obtain a dry crust, a considerable

* U. S. Patent, No. 563,709, July 14, 1896.

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quantity of it will be redissolved by the lead. It will rise again to the surface when the lead cools, but it will be rich in lead, and at the same time low in silver. It must, therefore, be returned

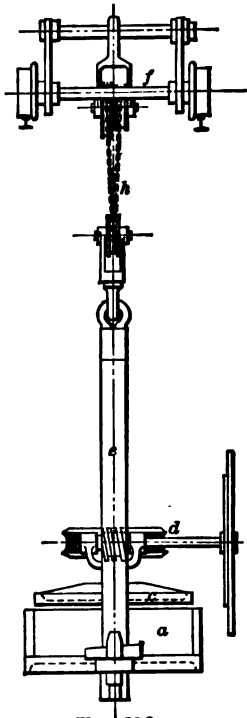
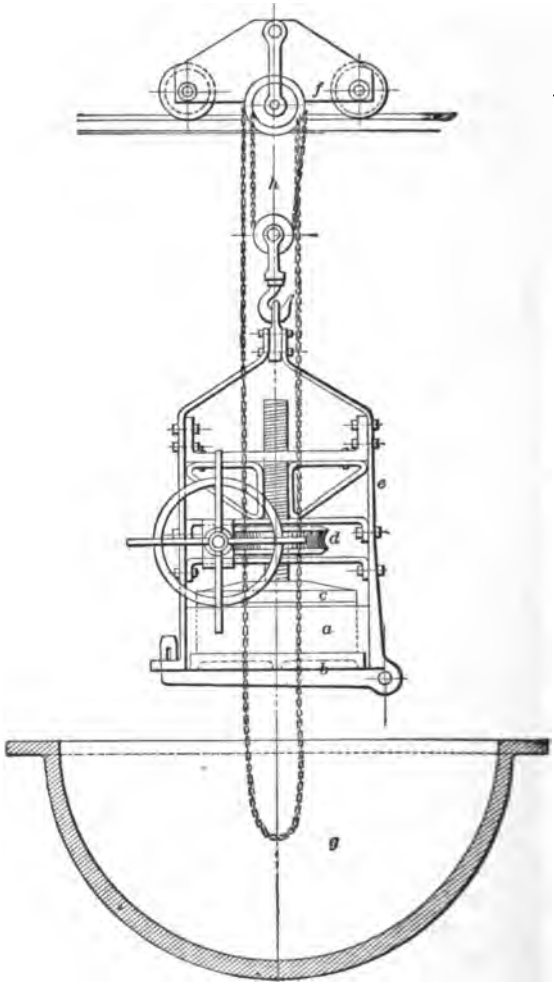


Fig. 352



FIGS. 377 AND 378.—THE HOWARD ALLOY PRESS.

to the next charge that is to be liquated, instead of going directly to the retort, as it would with any of the three apparatus just discussed.

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§ 110. Mode of Conducting the Desilverization.—This is as follows: To the lead in the kettle, which represents, after the kettle-dross has been removed, about 90% of the bullion charged into the softening furnace, is added lead obtained by liquating the gold crust from a previous charge. (With the Howard press this is not necessary.) The lead is then heated above the melting point of zinc (412° C.) and receives the first zinc to remove the gold and the copper. The heating takes a very short time; in half an hour from the time when the kettle is drossed the zinc will usually be melted down.

It is customary to place the slabs of zinc on top of the lead and to heat this until the zinc has become thoroughly melted, when the stirring-in can begin. Objections have been and are raised to this method of adding the zinc, which, floating on the lead, is exposed for some time to the oxidizing action of the air. To avoid this, the zinc has been inclosed in a perforated iron box, which is forced down to the bottom of the kettle and held there by an upright iron rod fitting into the center of a piece of flat iron which reaches across the kettle, and is fastened to it by set-screws. There can be no question that the zinc will melt more quickly at the bottom of the kettle, will not be exposed to the air, and rising upward in thin little streams will combine more readily with the lead and the silver, and will require less stirring-in than if melted while floating on the lead. French refineries use this method frequently. According to Edelmann and Rössler,* it is advantageous to melt the zinc in a separate vessel and to pour it into the lead, as it is then taken up quickly by the lead. It seems, however, doubtful whether less dross is formed than if the zinc is melted down floating on the lead. Figures comparing the results of these three methods of incorporating the zinc would be of interest.

In order to bring the melted zinc into intimate contact with the argentiferous lead, it is stirred in. When this is done by hand, a paddle is used, consisting of a perforated disk 12 in. in diameter, riveted to a handle 6 ft. long, having a cross-piece as hand-hold. Two men standing opposite each other do the stirring. They insert the paddles vertically at the rim of the kettle, push them downward toward the center, then raise them, using the rim of the kettle as a fulcrum, and draw them, with the

* *Loc. cit.*

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disks gliding on the lead, from the center toward the periphery, giving the lead a rotary motion, which they reverse every five minutes, thus insuring an intimate mixing of zinc and lead. This stirring-in, which lasts from one-half to three-quarters of an hour has always been hard for the workmen, and has been replaced at many works by steam-stirring, which is now giving way to the Howard mechanical stirrer (see below).

The effect of steam when introduced into lead containing zinc varies greatly according to the temperature of the lead.

1. If the temperature of the lead be below the melting point of zinc, *i.e.*, the temperature when the kettle is skimmed, the steam will bring to the surface a zinc crust, and with it some of the silver contained in the lead.

2. If the temperature be slightly above the melting point of zinc (stirring-in time), the steam will cause a thorough mixing of zinc and lead.

3. If the temperature be between a dark-red and an incipient cherry-red, the steam will cause a scum to rise, containing about 3% of zinc, which does not, however, take any silver away from the lead.*

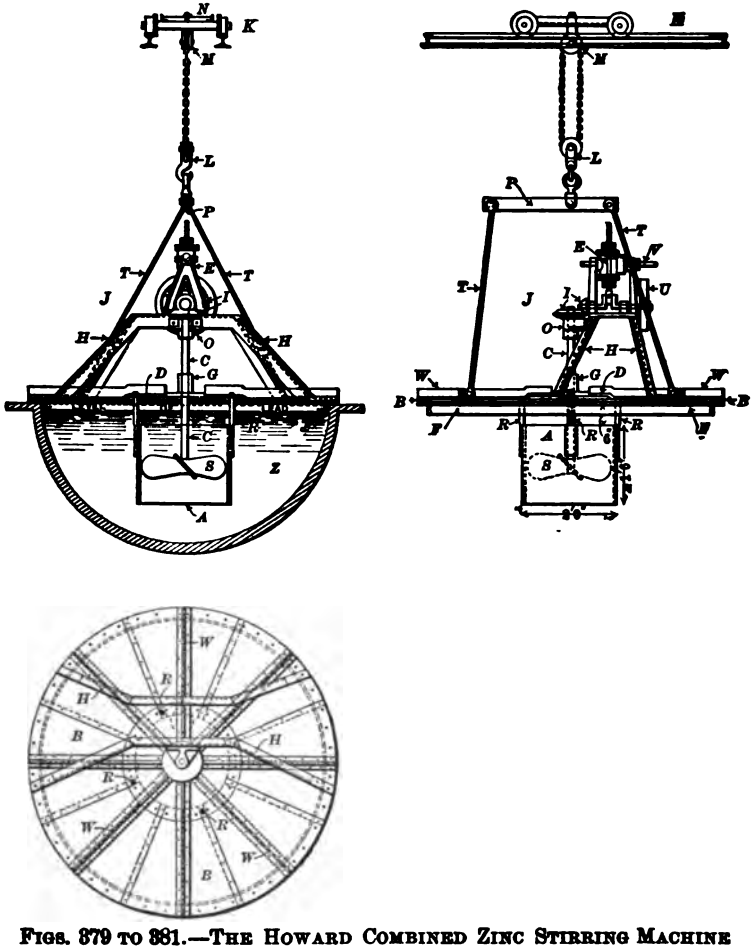
4. If it be a clear cherry-red, the zinc will decompose the steam; the resulting zinc oxide (mixed with lead oxide) collects as a powder on the surface of the lead.

The steam must be absolutely dry if violent explosions are to be avoided. The condensed water is separated by a steam trap placed beneath the working platform. To the vertical pipe coming through the platform is fastened by means of a coupling a small piece of pipe, to which is joined by two elbows (with a nipple intervening) the horizontal pipe, which reaches to the center of the kettle. An elbow connects it with the vertical pipe, that will reach 24 in. into the kettle when in place. Before the steam valve is opened, the vertical pipe is turned up, in order that the steam may first pass out into the air to warm the pipe and to insure the expulsion of any condensed water. The pipe is then turned down and weighted with a bar of lead to keep it in place. When the steam is turned on, the waves of lead caused by the first ascending bubbles will drive the zinc floating on the lead toward the rim of the kettle, and hardly any of it would

* Rösing, *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xxxvii., pp. 76, 77.

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become incorporated with the lead if it were not pushed toward the center of the kettle to be drawn into the lead by the downward current close to the steam-pipe. The zinc crusts that rise to the surface soon after stirring-in has begun are also pushed toward the center, that they may take up more silver. Thus the



FIGS. 379 TO 381.—THE HOWARD COMBINED ZINC STIRRING MACHINE AND COVER.

zinc and then the crusts pass down at the center, and come up again nearer the circumference of the kettle, whence they are again pushed toward the center.

The tool used for this purpose is a wooden hoe, consisting of

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an inch board, 12 by 18 in., into the center of which is inserted an inch lath from 8 to 10 ft. long.

The Howard "Combined Zinc Stirring Machine and Cover,"* represented in Figs. 379 to 381, is the only stirrer that has proved successful, the mechanical stirrers of Cordurié and others† having become obsolete many years ago. The device consists of three parts—the cover *B*, the stirrer *J*, and the traveler *K*. The cover *B*, made of sheet iron, is strengthened by T-irons *W*, and has an angle-iron flange *F*. When it is brought over the kettle *Z* and lowered to rest on the rim, the flange will sink into the lead and seal the surface. This prevents air from passing over the metal bath while the zinc is being stirred in. The air inclosed between the metal and the cover, being a bad conductor of heat, reduces the loss by radiation. From the cover is suspended by rods *R* a sheet-iron cylinder *A*, 2 ft. in diameter and 1 ft. 6 in. high, the top being 6 in. below the rim of the kettle and submerged in the lead. Inside of the cylinder is the propeller *S* attached to the vertical shaft *C*, which is supported by the collar *O*, and guided by the box *G*. The cover carries the two standards *H* holding a plate for the pedestals of the oscillating reversible steam engine *E*, having its steam inlet *V* on one side and the outlet opposite. The inlet is connected with the steam-main by a nipple and a hose. The engine, which is of three horse power for a 30-ton and of five horse power for a 60-ton kettle, rotates the propeller shaft through a double crank and bevel gear at the rate of 110 revolutions per minute. With an elliptical 60-ton kettle two propellers are used. By the action of the propeller the metal in the inclosing cylinder is forced out at the bottom, being constantly replenished from the top. Thus a perfect circulation and stirring-in, first of the zinc and then of the zinc crust that has risen to the surface, is obtained in the short space of from 7 to 10 minutes with a 30-ton kettle and in from 15 to 20 minutes with a 60-ton kettle. By reversing the engine the flow of metal will be in the opposite direction. As the air is almost completely excluded, the oxidation of zinc and lead is great reduced. The whole apparatus is suspended from a traveler *K* by means of the differential pully *M L*, the cross-piece *P*, and the four iron rods *T* bolted to the T-irons on the covers.

* U. S. Patent No. 529,617, Nov. 20, 1894.

† Grüner, "L'état actuel de la métallurgie du plomb," Paris, 1868, p. 72.

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The traveler runs on the same rails as the press previously described. The cost of stirrer and press is $500.

The method of working is simple. Assuming the kettle to be filled and drossed and thus ready to be heated for the first zinc addition, the cover is lowered, the kettle heated up, the cover then raised slightly for the addition of zinc and lowered again and the engine started, after the zinc is melted. After the stirring-in is finished, the cover and propeller attached to it are raised up high and pushed to one side. When the kettle has cooled, and the zinc crust has been removed, the cover is brought again over the kettle, lowered, and the metal heated up until it is ready for the next zinc addition.

The advantages of this method are that it saves time and labor, and that on account of the exclusion of the air and the better stirring, the consumption of zinc is reduced, the zinc crust made richer in silver, and very little dross and blue powder made in retorting. By the combined use of stirrer and press the zinc consumption with 300-oz. bullion has been reduced to 0.65%, the zinc crust made twice as rich, thus reducing the amount of retort bullion to be cupelled one-half and correspondingly increasing the yield of refined lead, and lastly the retort dross brought to an insignificant figure of from 10 to 12 lb. per ton of crust.

After the zinc has been stirred in, either by hand, by steam or mechanically, the fire beneath the kettle is removed or damped with wet slack coal, the fire-doors and ash-pit doors and the damper in the flue are thrown wide open, and the kettle is allowed to cool. At St. Louis, Meyer* tried to hasten the cooling by means of water-cooled pipes bent to the shape of the kettle, but his apparatus has found no favor, as the pipes did not cool the lead enough to pay for all the trouble and inconvenience connected with their use. At St. Louis they were given up many years ago.

Another method that has lately come into use is to blow cold air on to the surface of the lead. A pipe 4 or 5 in. in diameter is suspended vertically over the center of the kettle so as to be about 3 ft. above the surface of the lead. The cold air chills the lead in the center, which sinks to the bottom, while hot lead rises at the sides. This method is effective after the bulk of the crust

* *Mining and Scientific Press*, 1882, vol. xliv., No. 5; *Berg- und Hüttenmännische Zeitung*, 1882, p. 391.

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has been removed; it is claimed that it counteracts somewhat the tendency of the silver crust to adhere to the sides, thus reducing the necessity of scraping them after the bulk of a crust has been taken off.

Another means of hastening the cooling, although not much used, as it injures the brick, is to sprinkle water from a hose against the inside walls of the fireplace. If the zinc is stirred in by steam, its use will be found effective afterward in cooling.

After from two to three hours the lead has cooled down so far that the crust begins to adhere to the sides of the kettle. It is then removed with a skimmer, the disk of which, made of $\frac{1}{8}$-in. iron, is from 14 to 18 in. in diameter; and the handle, of $1\frac{1}{4}$-in. iron gas-pipe, is 7 ft. long, having a crosspiece as hand-hold. The work is facilitated by suspending the skimmer by a hook, which acts as a lever. Two men work together. One man pushes with a wooden hoe the crust toward his partner, who takes it up with the skimmer. Before discharging it into the liquating kettle, the mold, or the Howard press, it is important that the skimmer be well jerked several times, in order that as much lead as possible may be drained off into the kettle and the crust obtained become dry. Toward the end of the operation both men have to work very slowly and carefully to avoid pushing the crusts back again into the lead, which would retard the work very much. When the crusts have been removed from the surface, the alloy adhering to the sides of the kettle has to be brought to the surface, which is done by scraping them first with a chisel-pointed bar and then with a wooden lath. The blade of the bar is of steel, 2 by 4 in., and the handle of 1-in. round iron. Skimming and scraping are generally repeated twice, after which no more crusts will rise to the surface. It takes about an hour to perform this operation. When finished, the fire under the kettle is again started, and this heated to melt down and stir in the next zinc. The time required for heating up varies from one to two hours, according to the amount of lead (obtained from liquating zinc crusts) that is added to the kettle.

A sample is taken from the lead to see whether all the gold has been extracted and how much silver. It is well to cupel eight samples of an assay-ton each, and dissolve the silver buttons together to ascertain if all the gold has been taken out.

The gold crust obtained by the first skimming is collected in

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one of the liquating kettles or, if the reverberatory furnace is in use, in flat, slightly conical molds that have been whitewashed.

The crust obtained from the Howard press, being already liquated, is broken up as shown in § 111.

While the kettle is being heated up, liquated lead from the first silver crust of the previous charge is added (which becomes unnecessary with the Howard press), and when the lead is sufficiently hot, the zinc for the extraction of the bulk of the silver. The poor second silver crust obtained from the previous charge is used with third crust, if there was any, and fresh zinc. The operations are the same as for the gold crust.

The second and third crusts are also sometimes collected in molds or pressed, but oftener the skimmer is discharged on a whitewashed iron plate. There the crusts remain until needed, when one end of the plate is raised by block and tackle, and the two crusts are slid into the kettle together. Some works discharge only the second silver crust on the plate, while the third goes into molds. In order to reduce the amount of second crust, supposing a kettle to show 40 or more ounces silver per ton, it is advisable, after stirring in the necessary zinc with the Howard stirrer, to skim the crust forming on the surface before the kettle cools and put it through the press. It will be found to assay as high in silver as the main crust.

After each skimming, samples are taken, and ½ assay-ton is assayed for silver to check the progress of the desilverization. After the last silver crust has been removed, the assay should show 0.2 oz. silver per ton, or less if corroding lead is being made. Should it prove to be slightly higher, say 0.4 or 0.6 oz., the introduction of steam at the low temperature of the kettle after the last crust has been taken off will be effective in causing more silver-bearing zinc crust to be given off by the lead, as stated above. If the steam is used from one-half to three-quarters of an hour, the silver contents of the kettle will be reduced, and thereby generally an entire zincing saved.

In a 30-ton kettle four hours are allowed for each zincing if the silver is extracted by four additions of zinc. The first four hours include the time the softened bullion is being run into the kettle, the last four hours the time during which the desilverized lead is being siphoned into the refining furnace. When the lead comes from the softening furnace it is usually hot enough to melt

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the first zinc quickly, and the melting down of the last zinc and skimming of the final crust require but little time, so that the hour necessary to tap the softening furnace and empty the kettle can be included in the standard four hours. With a skilled crew it is possible to desilverize a 30-ton kettle with four zincings in 16 hours, but every moment has to be utilized to accomplish it. Some works, therefore, aim to extract the precious metals with three additions of zinc, and give five hours for each addition, keeping the gold and silver crust separate. When gold and silver are extracted together, a 45-ton kettle can be desilverized with three additions of zinc in the same length of time. The quickest work is that of a 60-ton kettle which is desilverized in two zincings, each lasting from 6 to 8 hours. The reduction of time may be still further increased by the substitution of oil or gas for coal as fuel.

The weight of the single crusts varies considerably, and also the amount of silver which they take up. As an illustration of stirring in by hand and liquating in a reverberatory furnace, 1,500 tons of softened bullion, containing 170 oz. silver and 0.5 oz. gold to the ton, gave 5% of liquated gold crust, required 0.3% zinc, and reduced the silver contents 15 oz. silver per ton; the liquated silver crust was 8%, required 1.5% zinc, and reduced the silver contents to 30 oz. per ton. The 1.5% represents the entire zinc added to extract the silver. The second silver crust reduces the assay of the kettle to about 3.0 oz. per ton, and the third to 0.2 oz. or less. Where only three zincs are added, the silver contents are reduced by the second silver-zinc from 30 oz. silver to the ton to 0.2 oz. or less. From the zinc added to the kettle part is recovered by distilling the zinc crusts, making the amount actually consumed lower. This was 1% for a year's run for softening bullion in the kettle, which averaged 150 oz. silver and 0.5 oz. gold to the ton, with the usual amount of copper. The coal required for desilverization, including liquation, is about 54 lb. for a ton of unsoftened base bullion.

With the Howard stirrer and press, the weight of the silver crust ready for the retort is 7.5% as against 13% with hand stirring and furnace liquation, the bullion assaying 300 oz. silver per ton and several ounces of gold; the total consumption of zinc is only 0.65% (a saving of 15%) and through saving of time the fuel is also reduced.

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In connection with desilverization may be mentioned an excellent custom* of breaking up unwieldy old kettles that have been set aside on account of leakage or corrosion. This is done by filling the kettle with water and suspending a dynamite cartridge from a floating board so that it nearly touches the bottom. When this is exploded the water will be thrown up into the air and the kettle broken into five or more pieces, which can be easily handled and shipped to the foundry.

§ 111. METHOD OF LIQUATING.—The liquation of the gold and silver crusts takes place, as stated in § 109, either in a liquating-kettle, a reverberatory furnace or the Howard press. In the kettle and the reverberatory furnaces it is essential that the temperature be raised gradually and that small quantities only be liquated at one time. If in the liquating kettle the temperature is raised quickly, some of the zinc crust will be taken up by the melted lead and carried off into the small liquated-lead kettle, whence it is again skimmed off as a crust rich in lead, to be treated with the next batch of crust. The kettle must therefore be large enough, as shown in Figs. 374 and 375, to hold the silver crust in not too thick a layer. With the reverberatory furnace a charge is introduced through the door near the flue, and gradually moved toward the firebridge, whence it is drawn out through the second door upon an iron plate let into the floor. By moving the crust from the coolest part of the furnace to the hottest, a gradual elimination of the lead takes place, and there is little danger of dissolving the crust, which grows less and less fusible, in the lead, which runs off as soon as melted.

With the Howard press, it is essential for the work to be done quickly, so as to prevent cooling. For the same reason the press must not be made too large, as if much time is needed to fill the press the crust and metal are liable to cool. A press 36 in. in diameter did not furnish as rich a crust as one 24 in. in diameter; with 60-ton kettles two presses are used.

The zinc crust, when still hot, can be easily broken up into small pieces. When removed from the kettle or the furnace, or dropped from the press, it is spread on an iron plate and reduced to nut size, and smaller, by flattening with the back of a shovel and working with a rake. In this form it is readily charged through the narrow neck of the retort. Another way to break

* Lautenthal Smelting and Refining Works. Private notes, 1890.

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up the liquated crust is to transfer it from the liquating apparatus into a cast-iron box (24 by 18 in., with a rim 3 in. high), the bottom of which has a series of slits (18 by 2 in.), alternating with ribs of the same width, the casting being ½ in. thick. With a liquating kettle and press this box requires a frame as support; with the reverberatory furnace it is attached to the door-frame.

The lead recovered by liquating in the kettle or the reverberatory furnace is from 40 to 60% of the weight of the crust charged. In desilverizing 250-oz. bullion, the lead from the gold crust assays from 100 to 200 oz. silver per ton, that of the first silver crust from 30 to 40 oz. The lead liquated from the gold crust is added to the desilverizing kettle before the gold-zinc is given, that from the silver crust before the first silver-zinc. With the Howard press, it, of course, flows into the same kettle whence it came.

§ 112. Refining Desilverized Lead.—The desilverized lead retains, after the last crust has been removed, from 0.6 to 0.7% zinc, according to the temperature that prevailed at the last skimming. To remove this, and also small quantities of arsenic and antimony that were either not entirely taken away during the softening, or that were introduced with the zinc used in desilverizing, the lead must undergo a refining process.

From the desilverizing kettle the lead is siphoned off into the apparatus used for refining, which is in most American refining works a reverberatory furnace, occasionally a spherical kettle.

The siphon (§ 115)is heated and filled by immersing it in the kettle, the stop-cock being open. When it has attained the temperature of the lead and is filled entirely with it, the stop-cock is closed with a key, the longer arm taken out and suspended, and the shorter one held down in the lead. The stop-cock is then opened, and the lead runs out into a cast-iron trough, which discharges into the refining apparatus. To keep the siphon in place it is weighted by a couple of bars of lead. Should the lead column break when the kettle has, for example, been half-emptied, and it not be possible then to fill the siphon again in the usual way, it will be necessary to invert and fill it by ladling; for this purpose an iron funnel may be used to avoid delay. But the breaking of the lead column is a very rare accident.

§ 113. Refining in the Reverberatory Furnace.—The general construction of the reverberatory furnace used for refining is the

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same as that for softening. Formerly it was customary to make the refining furnace smaller than the softening furnace, in order that it might correspond to the smaller amount of lead it had to treat. For instance, the hearth of a softening furnace of a 25-ton kettle was made 13 by 9 ft. and 13 in. deep, having 13-in. side-walls; that of the refining furnace 12 by 8 ft., and of the same depth, with the same thickness of walls. At present it is commoner to give the refining furnace the same dimensions as the softening furnace, only making the hearth slightly shallower, thus simplifying the iron parts. If in the softening furnace, shown in Figs. 340 to 346, the depth of the hearth is reduced 3 in., it will have the capacity necessary to receive the desilverized lead from the kettle, which will fill it to just below the skimming doors. In some refining furnaces the lowest point of the hearth is placed beneath the flue, as in the softening furnace described; in others, below the central skimming door. By referring to the general plan (Fig. 338) it will be seen that the lead is discharged from below the door next to the flue. The arrangement for tapping is usually the same in both furnaces. If, however, the refined lead is to be conveyed directly from the furnace into the molds, as is still found in a few instances, instead of being molded from the "Merchant-kettle," the tap will be slightly different from that of the softening furnace (see § 115).

The mode of operating is similar to that in the softening furnace. When the furnace is filled, the fire is urged, as a high heat is required to burn off the zinc. This is partly volatilized and carried off with the fumes, and partly oxidized and scorified by the litharge which forms at the same time. Some refiners add lime to the charge, as in softening. After heating about four hours the surface of the lead will be covered with a heavy litharge-like skimming. The doors are thrown open, the skimming is removed, and the second heat given, after which cooling and skimming are repeated. A third heat is often necessary to slag the last traces of zinc and antimony. When these are completely removed, the litharge drawn from the surface of the lead by means of a rabble should be in large, thin plates. It should, while hot, have a bright-yellow color when seen in bulk, and a greenish-yellow when held up to the light, but not one brown spot (antimony) should be visible. If these large flakes of lith-

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arge should become dark or show spots after having been exposed for some time to the air, the lead is not sufficiently refined to satisfy the requirements for corroding-lead.

A bar of lead molded at the correct temperature should show on the surface fern-like crystalline forms, the branches at right angles to the main axis. If the lead is not sufficiently refined, there will form on the surface in addition to the fern-like crystals, small bunches of crystals similar to snowflakes, which disappear again when the bar solidifies completely, leaving the surface finely-crystalline. A ladle filled and emptied two or three times with lead will show in the bowl the blue color of a bar of refined lead, if all the impurities have been removed. Lastly, a bar of lead will often show on the lower side of the lip a blue color, if the lead has been satisfactorily refined, although the absence of this color will not prove that the lead is of inferior quality.

In firing the refining furnace it is essential for the temperature to be raised quickly to the necessary intensity, and kept there. If it is allowed to fall even a little, the burning off of the zinc will be greatly retarded, and with it the dependent operations.

In order to hasten the operation, steam is used. In addition to the mere mechanical effect of stirring, it acts chemically by being decomposed by the zinc (§ 110).

It is introduced either through injectors in the roof and in the sides near the bridge or through pipes inserted into the lead through the doors so as to nearly touch the hearth. With the pipes the steam pressure is correct when the lead is thrown up about 1 ft. In order to obtain skimmings thoroughly melted and free from lead it is essential to keep the temperature high; if it is too low the skimmings are liable to be powdery and filled with shots of lead. Steam, if rightly used, will not increase the amount of skimmings.

The time required for refining with a 30-ton plant, without the use of steam, ranges from 10 to 15 hours; with four steam-jets it is four hours; with six jets three hours. The refining skimmings amount to from 4 to 5% of the bullion charged, and contain about 90% of lead. The coal consumed in refining is about 120 lb. per ton of unsoftened bullion, corresponding to about 7 gal. of reduced oil, air being used as atomizer.

§ 114. Refining in the Kettle.—The second method of refining desilverized lead is the one invented by Cordurié, who intro-

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duced steam into the lead heated up to cherry-redness, the oxidized zinc collecting in the form of a powder on the lead. As the air cannot be excluded from the surface, and as it is also carried in by the steam, some of the lead is oxidized, and the pulverulent yellow mass floating on the surface consists of a mixture of lead and zinc oxide and finely divided shots of lead. The composition of these oxides when taken from the kettles at the Lautenthal* Smelting and Refining Works (Prussia) is Sb_2O_5, 1.893%; Fe_2O_3, 0.986%; ZnO, 23.775%; PbO, 37.933%; Pb, 34.236%.

The base bullion at Lautenthal is desilverized without previous softening; hence the high percentage of antimony.

An analysis of the oxides taken from the refining kettle of the Pueblo Smelting and Refining Co. gave, according to Howard: ZnO, 12.5%; PbO, 6.0%; Pb, 81.0%.

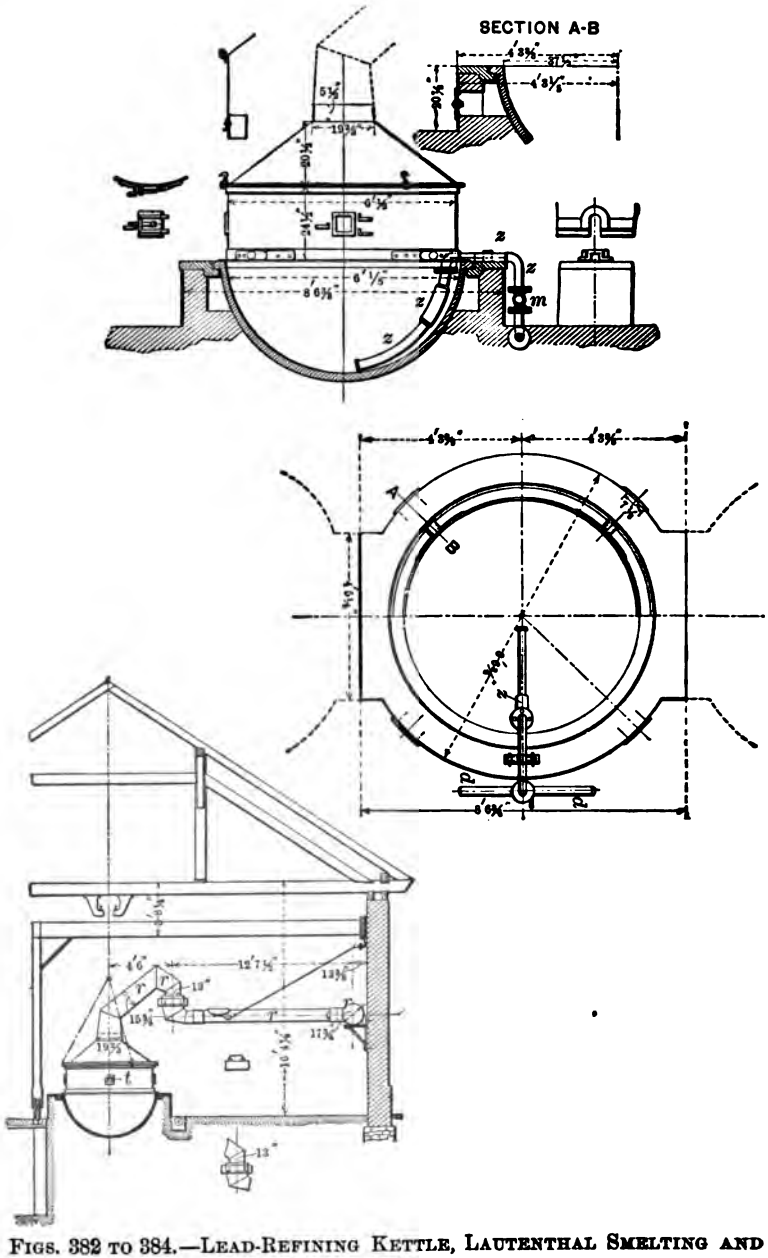
The larger shots of lead of the oxides at Lautenthal are separated by screening, the finer ones by washing over an inclined plane. The impalpable powder, forming 15.44% of the whole, floats off and is settled in vats, dried in a reverberatory furnace, and forms a reddish-yellow paint of good covering power. It consists of 60 to 67% ZnO, and 33 to 40% PbO. The residual shots of lead are smelted at intervals for a second-class lead, as they contain some antimony.

At Lautenthal† (Figs. 382 to 384) the cast-iron kettles (6 ft. in diameter, and 3 ft. 2 in. deep, holding 12½ metric tons of lead) are heated, after the desilverization is finished, in four hours, to a cherry-red; superheated steam having a pressure of from 29 to 36 lb. per square inch is then introduced through a cast-iron pipe *z*, bent to the form of the kettle, so that the steam enters at the bottom. After two hours all the zinc has been oxidized. In order to decrease the loss of heat by radiation, to keep off the air, and to prevent the oxides from being lost, the kettle is covered by a movable sheet-iron cylinder, which has near the lower rim two opposite doors (4¾ in. square) and the opening for the steam-pipe. It ends in a conical hood which leads the vapor and dust through a sheet-iron pipe into the main flue, terminating in a dust chamber. The cylinder with its hood

* *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xxxviii., p. 272.

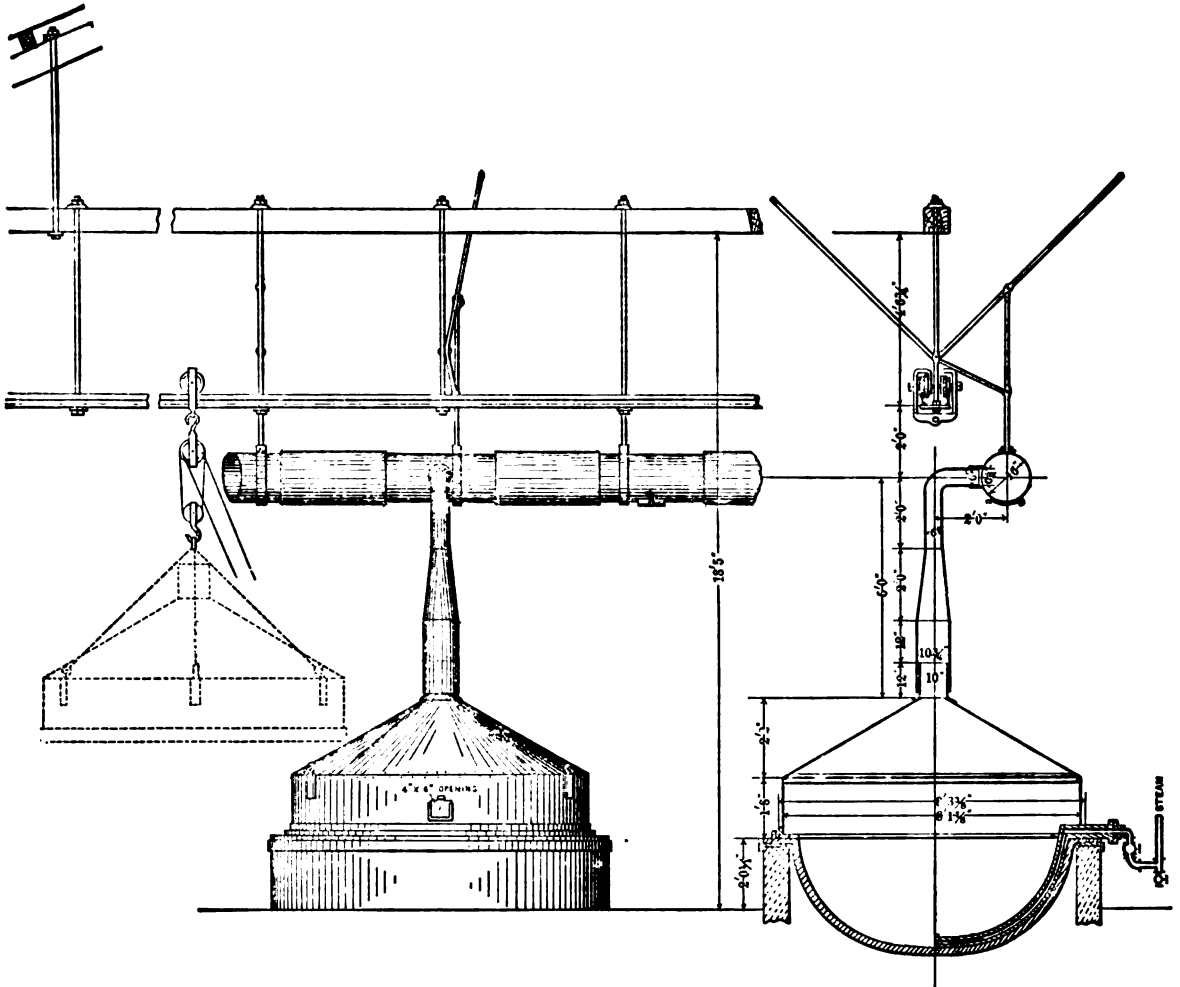
† Private notes, 1890; Illustrations from Schnabel, "Metallhüttenkunde," Berlin, 1894, vol. i., p. 444.

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FIGS. 382 TO 384.—LEAD-REFINING KETTLE, LAUTENTHAL SMELTING AND REFINING WORKS.

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FIGS. 385 AND 386.—LEAD-REFINING KETTLE, PUEBLO SMELTING AND REFINING CO.

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and pipe is suspended by a running differential pulley from a traveler. For every 100 lb. of unsoftened bullion 4.67 lb. of paint are produced, which is higher than the percentage of skimmings in the reverberatory furnace, as the paint forms but a small part of the total lead taken out of the kettle.

Part of the steam refining plant of the Pueblo Smelting and Refining Co., Pueblo, Colo., is shown in Figs. 385 and 386. The kettle, 8 ft. $1\frac{7}{8}$ in. in diameter, and 3 ft. deep, is intended to treat 20 tons of desilverized lead. Its rim has two concentric rings forming ridges, between which is a groove to receive the bottom of the hood. The air can then be entirely excluded by sealing the joint with sand or oxides. The sheet-iron hood is raised and lowered by means of a differential pulley suspended from a traveler and joined by a vertical pipe to a suspended horizontal main leading into the dust chamber. The dimensions of these pipes are smaller than those of the pipes at Lautenthal, because the former serve only two kettles, while the latter are intended for a larger number. It takes three hours to heat up the kettle, and two hours to steam it, about 1,700 lb. of oxides being produced.

If the kettles were made larger, there would be danger of the bottom becoming weak in heating up the lead to the required temperature, and yielding to the pressure in steaming.

For the success of the operation it is essential that the lead be kept at a cherry-red, as then the zinc, and incidentally some lead, will be quickly oxidized and float on top of the lead in the form of a fine powder readily removed with a skimmer. If the temperature sinks, the time of steaming is lengthened and the oxide formed will be mushy and retain a large quantity of shots of lead disseminated through it. Kuhlemann* gave the following facts as to the elimination of zinc: After steaming ½ hour the lead retained 0.402% zinc, after 1½ hours 0.116 %, the zincy lead still showing its characteristic of flattening out readily; when this had disappeared and the oxides had assumed the form of a dry powder, *i.e.*, after two hours, the lead retained 0.0004% zinc. The operation is finished when the oxides are powdery, and a sample of lead taken up in a ladle forms clean golden-yellow litharge whenever its surface is exposed to the action of the air.

* Private notes from the Lautenthal "Record of Analyses," 1879.

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The oxides are usually removed with a perforated skimmer, which is a lengthy and hot piece of work. The operation is very much simplified by the use of the Howard skimmer,* shown in Fig. 387. This was originally constructed to remove the zinc

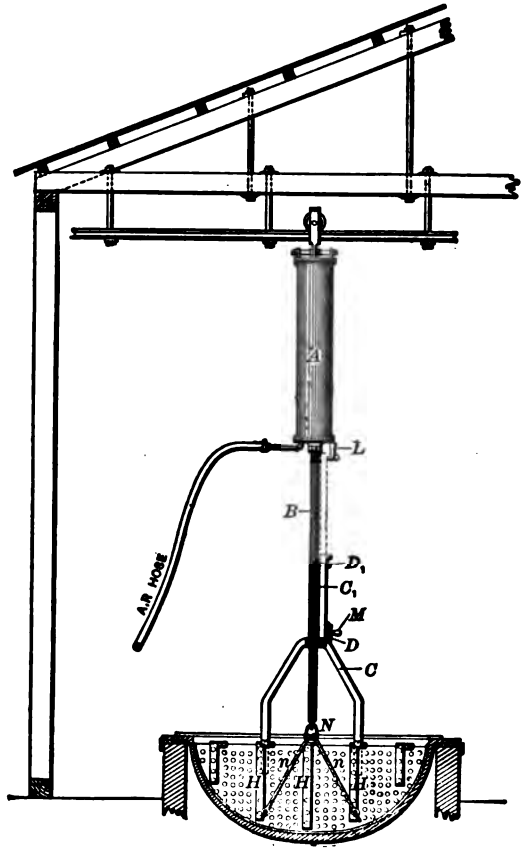


FIG. 387.—THE HOWARD SKIMMER.

crusts from desilverizing kettles.† It has been replaced in most instances by the Howard press, and is now used for oxides. It consists essentially of two parts: a yoke *C* with slotted extension *C'*, carrying a pair of hinged skimmer-leaves *H*, of perforated

* U. S. Patent, No. 502,122, July 25, 1893. † "The Mineral Industry," ii., p. 440.

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boiler iron, and a suspended air-cylinder *A*, with piston-rod *B*, carrying a link *N*, with two pairs of rods *n*, one for each of the leaves. The piston-rod carries a pin *D*, traveling in the slot of the extension *C'*, and the extension has a lock *M* to make a rigid connection between the yoke and the piston-rod; the air cylinder has on the lower side the air inlet and opposite to it a latch *L* to secure the projection of *C'*.

Supposing the kettle to be ready to be skimmed, and the skimmer to be immersed in the lead, as shown in the figure, compressed air will be turned on through the hose. This causes the piston-rod to rise and the leaves to unfold. The yoke remains stationary, and extending into the lead prevents the leaves from turning. When the piston-rod has opened the leaves, the pin *D* will have moved up in the slot *C'* to the position *D'*, when it engages with the yoke extension and pulls up the skimmer with its load until the projection of *C'* is latched by *L*. It is now pushed to one side, where the oxides are to be unloaded. For this purpose some air is let out of the cylinder, when the leaves will drop, the yoke remaining latched by *L*. When the skimmer is to be used again, it is brought over the kettle, the yoke locked with *M* to the piston, the latch *L* raised, and air let out again, when the closed skimmer will sink into the lead.

The use of compressed air makes the work very easy. If sufficient pressure cannot be had, the skimmer is worked by hand, substituting for the air cylinder a combination of rack and other gearing.

The great drawback of this method is the wear and tear in kettles. According to Schmieder,* at Tarnowitz, where the lead is very free from antimony, cast-iron kettles hold out only for 20 charges, while steel kettles are good for 90 charges. The life of cast-iron refining kettles varies greatly; extreme figures are 30 days and 120 days. Of kettles from the same foundry, cast under apparently the same conditions, one within the writer's experience will last only a month, while another lasts over a year. The life of a kettle is greatly prolonged if it is used exclusively for dezincifying lead, and has not to serve, as is the case in most European works, both for desilverizing and refining.

As by the use of steam a considerable amount of air is carried into the lead and some of this oxidized, Rössler† tried to replace

* *Berg- und Hüttenmännische Zeitung*, 1887, p. 377. † *Ibid.*, 1890, p. 243.

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it by different gases. With carbon dioxide, the lead being heated to 700° C., in a short time all the zinc was converted to white oxide, and could be skimmed off from the lead. As carbon dioxide cannot be easily obtained pure to be used for such a purpose, he tried a mixture of carbon dioxide and nitrogen, obtained by pressing air through a cylinder filled with glowing coal. The result was a gray powder, in which some of the zinc was present as metal in a finely divided state. By the use of a mixture of carbon monoxide and nitrogen, drawn from a gas-producer, a powder of a darker gray was obtained, containing up to 75% of its zinc as metal in a finely divided state, the rest being oxidized by the carbon dioxide present. The refined lead was entirely free from zinc if the temperature was kept above 700° C., otherwise a zinc crust formed on top of the lead. In working on a large scale with gas containing from 15 to 20 volumes of carbon

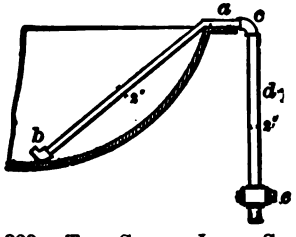


FIG. 388.—THE STEITZ LEAD SYPHON

dioxide, too many difficulties were encountered, according to Schnabel,* to make the process a practical success.

§ 115. MOLDING FROM THE REFINING FURNACE.—The molding of lead was formerly universally done by ladling it from a kettle. This has, however, become antiquated. In some works the lead is molded directly from the reverberatory furnace in which it has been refined. At the lowest part of the furnace, on the side away from the desilverizing kettle, a piece of 2-in. gas-pipe is screwed into the pan if of cast iron; if of wrought iron, into the flanges placed on either side and fastened together with bolts. If the furnace has a wrought-iron jacket, the pipe *b* (Fig. 389) is screwed into the cast-iron plate placed between the two sides of the jacket at the tap-hole. The pipe *b* is joined by a cast-iron stop-cock *c* to a T, whose horizontal arm is closed

* "Metallhüttenkunde," i., p. 441.

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with a plug *d*. To the vertical end of the T is attached a nipple *e*, with an elbow *f* at the lower end, into which is screwed a long pipe *g* (from 7 to 10 ft.), which can be moved horizontally by the arm *h*, while it discharges the lead into molds placed in a semicircle, the center of which lies beneath the nipple. The molds commonly used now (Fig. 390 to 392) differ from the ordinary

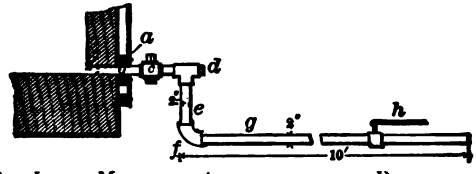
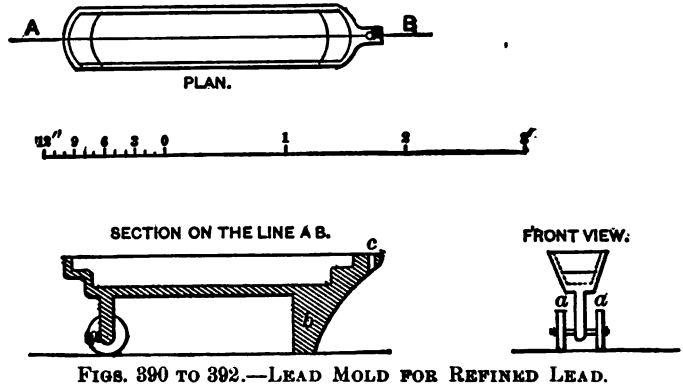


FIG. 389.—LEAD MOLDING APPARATUS FOR REFINING FURNACE.

blast-furnace molds in that one end rests on two wheels *a*, while the other has a leg *b*. The lip of the mold above this has a hole *c*. By passing a hook through it and tilting the mold, it is run away, the bar tipped out, and the mold then quickly returned to its former place. At some works the molds are made large enough to hold three bars of lead. The lip then has, instead of



FIGS. 390 TO 392.—LEAD MOLD FOR REFINED LEAD.

the hole *c*, a rectangular socket running horizontally, into which is inserted a slightly bent iron handle to move and tilt the mold.

When the furnace is to be emptied a charcoal fire is started under the stop-cock, and the horizontal pipe immersed in the lead of the refining furnace to be warmed. It is then screwed into the elbow, the stop-cock is opened, and the lead run into

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the first mold of the semicircle. This warming of the pipe is, however, not necessary. If the stop-cock be opened entirely, the first lead arriving at the end of the horizontal pipe will still be liquid. After that, the cock will have to be slightly closed, as the molds would otherwise fill too quickly for the man, who has also to attend to the skimming of the surface of the bars. This he does with two thin pieces of board the width of the mold, collecting the dross between them and dropping it on the floor. Another method is to rake off the dross on to the floor with a bent piece of hoop-iron. The former method gives a cleaner bar.

One mold after another is thus filled. When the lead in the first three or four molds has solidified, it is chilled with water, the pigs are trimmed with a sharp, chisel-pointed bar, and the molds run off to the wall of the lead-pit (Fig. 338), where the lead is to be dumped before weighing, and brought back empty to their places. In this way from 40 to 60 molds that form the semicircle are filled one after the other.

This method of molding has the advantage that the vertical distance between the shipping level of refined lead and the receiving level of base bullion can be less than with the methods to be discussed; then the filling of the molds need not be continuous, as it must with the siphon. With a kettle holding 30 tons of lead, the molding can thus be given in contract to the four men in addition to the charging, softening, desilverizing, and refining. They do it in two separate operations, molding part of the lead between the second and third zincings, and the rest while the last crust is rising, the total time required for molding being six hours. The method has the disadvantage that the furnace has to be cooled down considerably for the lead to attain the right temperature for molding. Thus, after every molding, two hours or more are required to heat up the furnace again for the next charge. This cooling and heating-up of the furnace with every charge cannot be good for the lining, therefore the molding of the lead directly from the refining furnace has not found so much favor as might be expected.

§ 116. Molding from the Merchant Kettle.—It has become more common to tap the refined lead from the reverberatory furnace into a kettle—the merchant kettle—(Fig. 338) heated from below, and to let it cool there till it has attained the correct

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temperature for molding. In this case the tapping of the furnace is done in the same way as from the softening furnace.

If the desilverized lead is refined by means of steam in a kettle, the molding is done either from the refining kettle or the lead is siphoned off into a merchant kettle below, to store the lead until it is time for molding, and thus have the refining kettle ready for another charge of desilverized lead.

Some refining works pole the lead in the merchant kettle at a low temperature, under the impression that an especially fine grade of corroding lead is thus obtained. Poling is, however, not necessary, as all the impurities not only can be, but ought to be, completely removed in one operation, be it in the reverberatory furnace or in the refining kettle.

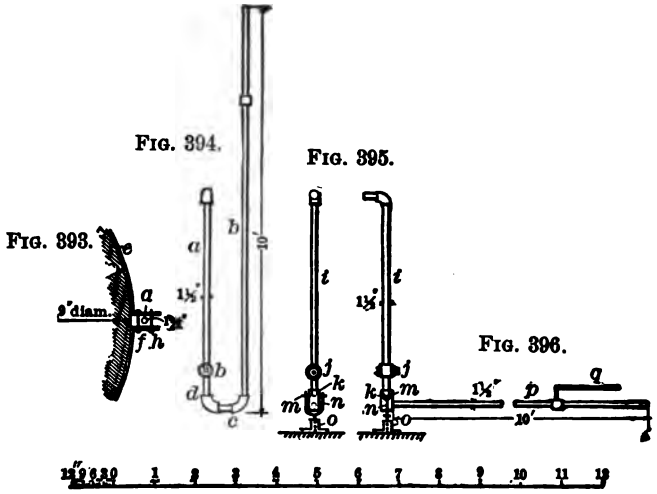
As this poling at a low temperature is the best method of purifying the otherwise pure leads obtained from smelting the clean, non-argentiferous ores of the Mississippi Valley, a few remarks are in place. The gases and vapors from the wood stir up the lead and expose continually new surfaces to the oxidizing action of the air. Thus the small amounts of arsenic, antimony, copper, zinc, and iron are slowly oxidized and collect on the surface as a dross. A crutch serves to keep the stick of wood horizontally depressed in the molten lead. It consists of a piece of flat iron long enough to reach about 1½ ft. over the rim of the kettle, upon which it is placed, and weighted with a couple of bars of lead on either side. To it are riveted two arms, say 2 ft. 6 in. long and 2 ft. apart, forked at the ends which, reaching into the lead, receive the wood; they are connected halfway down by a cross-piece of flat iron. If the lead is to be poled with a billet of green wood, this and the crutch are best put in place before the kettle is filled, as thus the tedious work of depressing a green stick of wood into hot lead is avoided. It is much simpler and just as effective to introduce steam near the bottom of the kettle by means of a 1-in. pipe bent to the form of the kettle and passing downward through the lead.

The molding of the lead from the merchant kettle in American refining works is almost always done with the Steitz siphon.

Two such molding apparatus are shown in Figs. 393 to 396. In Figs. 393 and 394, *a* represents the longer arm of the siphon, made of 1½-in. pipe, with the cast-iron stop-cock *b*. At the lower end it is joined by two elbows, with a nipple intervening, to the

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swinging-pipe *b*. This can be turned down around the center *c*, and when in that position it can be moved in a horizontal circle having its center at *d*. The siphon is filled just like the one shown in Fig. 388, with the exception that the handling is done with the swinging pipe *b*, instead of with a pair of tongs. As in molding, the swinging pipe has to describe nearly a semi-circle; the ordinary way of keeping the siphon in place by weighting with a couple of bars of lead is not sufficient. For this purpose two iron hoops about 2 ft. 3 in. apart pass around the brick-work of the kettle. The two ends of a hoop are bent,



Figs. 393 to 396.—Lead Molding Apparatus for Merchant Kettle.

as shown in Fig. 393, and tied by a bolt. The vertical arm of the siphon *a* is held in place between the two hoop-ends, the fixed bolt *f* and the movable one *h*.

Another arrangement for molding is shown in Figs. 395 and 396: *i* represents the longer arm of the siphon with the stop-cock *j*; it discharges the lead into a 3-in. pipe *k*, closed at the bottom. This has two trunnions *m*, which swing in the bearings *n*. They are joined to a pivot rotating in the socket *o*. The pipe *k* is connected with the swinging arm *p*, which is moved with the handle *q* over the molds. At some works the bottom of pipe *k* is closed with a cap having a socket, into which the pivot, fastened

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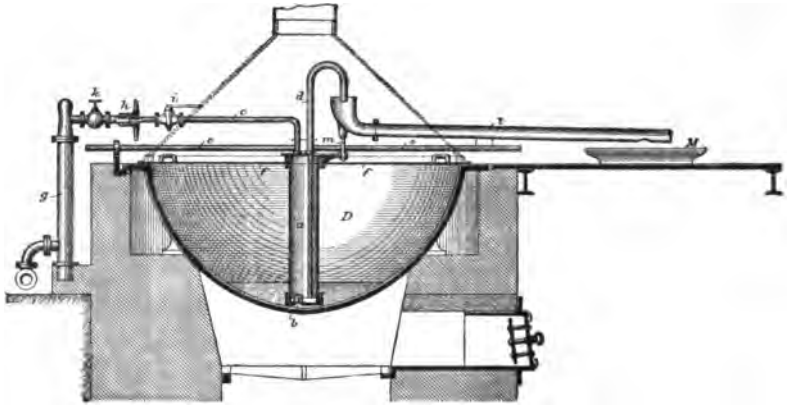
to the floor, fits loosely. Thus a number of slight variations in detail are found at different works.

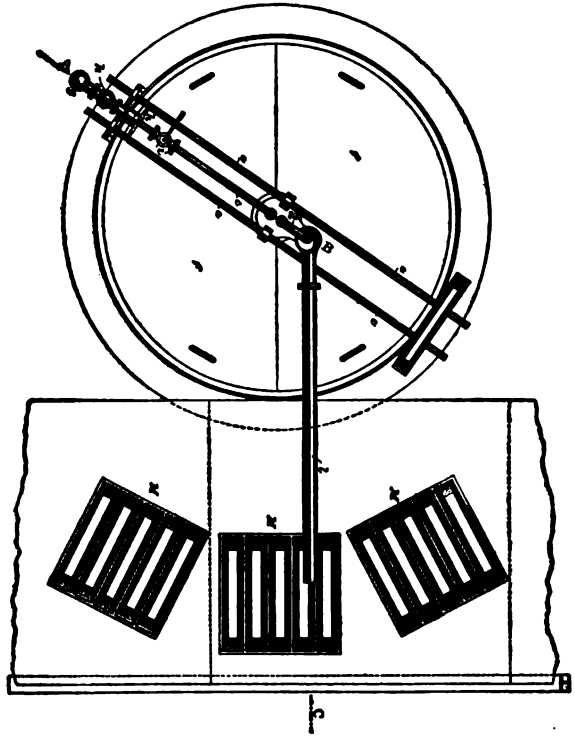
In the older European works, generally erected on level ground, the Rösing lead-pump* has come into universal use, and is doing most satisfactory work. It is found also in this country. Figs. 397 and 398 show its latest form. It consists of a cast-iron cylinder *a* closed at both ends, except that in the bottom it has a globe-valve *b* and in the top two openings, one for the steam-pipe *c*, which simply enters the cylinder without extending any distance into it, the other for the lead-discharge pipe *d*, which reaches nearly to the bottom and rises well above the cylinder, delivering the lead into the swinging-pipe, pivoted at *m*. The pipe *c* has a three-way cock *i*, so as to connect the interior of the cylinder with the steam-pipe and the open air. It is joined to the branch-pipe *g* by an expansion coupling *h*; *k* is a globe valve. The pump is held in position by two iron rods *e*, which pass through eyes in the top of the cylinder. Their ends are secured at one side of the kettle by lugs, at the opposite side by bars of lead. The kettle *D*, while discharging, is covered with a lid *f*, two semicircular pieces of sheet iron.

When a kettle is to be emptied, the cylinder, first warmed by floating on the bath of lead, is lowered into it, with the three-way cock set so as to connect the interior with the air, and secured; the kettle is covered with the lid and the steam connection made. In lowering the cylinder, the lead rises on the inside as well as in the discharge-pipe. The cock is turned to close the air channel and open the steam channel, the steam valve is opened and steam of from 30 to 37 lb. pressure (from 15 to 22 lb. in excess of atmospheric pressure) admitted. This will close the globe valve *b*, and forcing the lead up and through the discharge pipe will empty the cylinder. When the lower end of the discharge-pipe has been uncovered, the steam will pass through it into the open air, the pressure will sink from 30 or 40 lb. down to 15 lb., and the lead outside of the cylinder will again raise the globe valve and enter the cylinder until the lower end of the discharge-pipe has been sealed, when the steam pressure will increase, close the globe valve, and force the lead up again and through the discharge-pipe, and so on. These alternate opera-

* *Engineering and Mining Journal*, Nov. 28, 1885; *Berg- und Hüttenmännische Zeitung*, 1889, p. 262; *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, 1893, xli., p. 294,

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FIGS. 397 AND 398.—THE ROESING LEAD PUMP.

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tions are repeated 30 times a minute, giving a continuous though not perfectly smooth stream of lead, there being a slight pulsation. At Tarnowitz, Silesia, 8¼ tons of lead are pumped to a height of 4 ft. 4½ in. every hour with a steam pressure of 35 lb. A drawback of the pump is that the lead from the ordinary hemispherical kettle cannot be entirely removed, there remaining about 800 lb. It has the advantage over the siphon that deeper kettles can be used.

§ 117. LABOR, FUEL, OUTPUT OF LEAD.—In many refineries the working of the softening furnace, desilverizing kettle, and refining furnace is given in contract as a whole, instead of having separate men working by the day at each furnace. Three men with a good head man can, if the work is well systematized, do everything that is necessary with a 30-ton plant. If the lead is molded straight from the refining furnace, it is possible for them to attend to it also, but this is almost to overwork them. Therefore it has become the general custom to store the lead in the merchant kettle and to mold from it. A separate contract is made for the molding and loading into cars. This is often taken by a set of men who unload, weigh, and sample the base bullion, and deliver it at the softening furnaces. They also move the base bullion and lead produced at the blast furnaces or the different reverberatory furnaces. By having a good head man for each desilverizing plant and one or two contractors for handling the raw material, the by-products and the market lead, the labor in the refinery becomes very much simplified and cheapened. With the use of the Howard stirrer and press there is a reduction of from 25 to 50% in labor.

The fuel consumed in softening, desilverizing, liquating, refining, and molding is about 330 lb. of soft coal per ton of base bullion.

The amount of lead recovered in the form of market lead varies somewhat according to the purity of the base bullion. It is about 80% of the bullion charged, or 88% of the softened lead in the kettle. This figure has been raised to over 85% by the use of the Howard stirrer and press.

§ 118. TREATMENT OF ZINC CRUSTS.—The working of the zinc crust has been and still is the weak point of Parkes' process. Many methods* have been tried, but only few survive. They

* Kerl, "Grundriss der Metallhüttenkunde," 1881, p. 314; Schnabel, "Metallhüttenkunde," Berlin, 1894, vol. i., p. 549; Roswag, "Désargentation de plomb," 1884., p. 296.

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are all based on the volatility of the zinc and the readiness with which it is oxidized. Two only will be discussed.

§ 119. FLACH'S PROCESS.—This consists in smelting the zinc crust in a blast furnace with a large percentage of slag, some matte and fluxes, the slag aimed at being ferruginous and low in silica, and the pressure of the blast not exceeding ½ in. mercury. The zinc is partly taken up by the slag and matte; a large part of it passes off into the dust chambers; the resulting rich bullion retains but little. If the crust contains any copper, it is taken up by the matte. In smelting zinc crust with 120% of slag, 30% of matte, 11% of fuel, and puddle cinder as iron flux, in a small 36-in. circular blast furnace with an Arents siphon-tap, the writer found that after two days' running a mushy substance collected on top of the lead, that refused to be taken up either by slag or matte. In order to keep up a good communication with the lead in the crucible, the soft mush had to be repeatedly removed from the front of the furnace. It seems, therefore, that if in exceptional cases it should be necessary to smelt zinc crust, it would be advisable to fill up part of the crucible with brasque and to tap the lead and matte, when the mush will be carried out of the furnace.

While there is no doubt that smelting the zinc crust in the blast furnace furnishes quickly a rich bullion to be cupelled, the process has been abandoned as a regular method of treating the crusts, as none of the zinc is recovered, and the great losses of silver and lead are made up only in a very small degree by resmelting the zinc-bearing by-products.

§ 120. DISTILLATION OF ZINC CRUSTS.*—This process, first used by Parkes, has undergone many improvements and has become the one universally used in the United States since Balbach first used graphite retorts. The method, therefore, often bears his name. Retorting in vacuo has been suggested, which would permit carrying out the process at a much lower temperature than is the case under atmospheric pressure, but so far it is merely a suggestion.

§ 121. FURNACES.—A furnace, to be suited for the process, must permit the raising and sustaining of a high temperature,

* Eilers, "Transactions of American Institute of Mining Engineers," iii., p. 314; Rösing, *Zeitschrift für Berg-, Hütten- und Salinen-Wesen*, xxxiv., p. 91; *Berg- und Hüttenmännische Zeitung*, 1886, p. 421.

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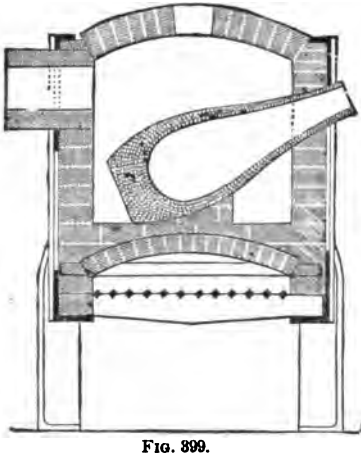


FIG. 399.

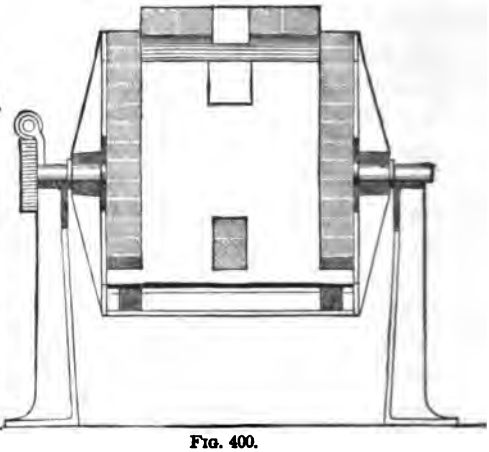


FIG. 400.

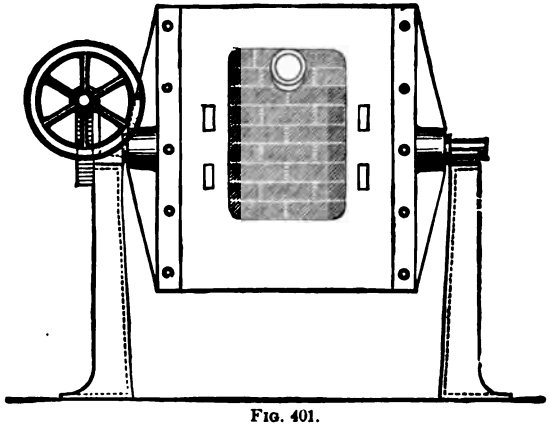


FIG. 401.

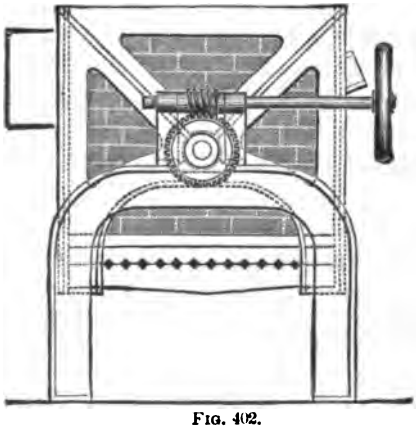


FIG. 402.

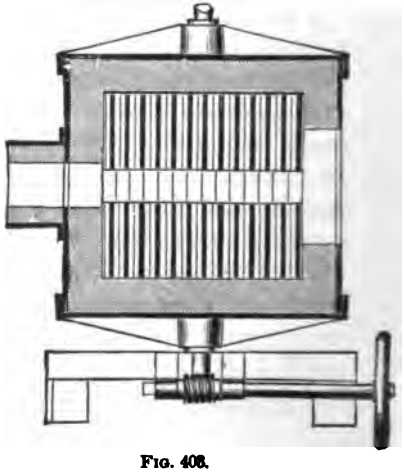


FIG. 403.

FIGS. 399 TO 403.—THE FABER DU FAUR RETORTING FURNACE.

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and at the same time be of such a construction that a broken retort can be readily exchanged, and the rich bullion that has run out, quickly and completely collected. Of the different forms three may be cited.

1. *The Faber du Faur Furnace* (Figs. 399 to 403).—This is a crucible furnace of cubical form, built into cast-iron framework that swings on trunnions, in order that the furnace may be turned over and the contents of the retort emptied. The furnace is closed at the top by an arched roof, which usually has one opening, the charging opening for the coke; the products of combustion pass off through a flue, which is generally placed at the back, as in the drawing, but sometimes at the side and occasionally in the roof. At the front is an opening for the neck of the retort. The bottom is formed by two sets of rectangular wrought-iron grate bars placed on edge. The retort rests on a small brick pillar, which is supported either by a brick arch (as in the figure) or by an iron plate running from front to back, and protected from the heat by two courses of firebrick. The rotation of the furnace is effected by a worm-gear, sometimes simply by means of a lever. The furnace in its original form, as represented by the drawing, was about 4½ ft. cube, and was intended for a pear-shaped retort holding 250 lb. of liquated crust. It was lined with a full course of brick, except at the front, where the brickwork was 4½ in. thick. At present the retorts, while retaining their original form, are made larger and thinner; they hold from 1,000 to 1,500 lb. of crust; the furnace has retained its original size, but is lined on all sides with a half course of brick. The old retorts (Fig. 399) were made of raw and burned clay mixed with about 25% of graphite to protect the clay from the corrosive action of the lead, and were very thick at the bottom; at present they are made of graphite to which sufficient clay (45%) has been added to give strength and stability to the retort. A 1,000-lb. retort is 36 in. high, 8 in. wide at the neck, 18 in. at the belly, and 13 in. at the bottom. It is 1⅛ in. thick at the neck, and increases to 2 in. at the bottom. A so-called 1,500-lb. retort is shown in Fig. 404. It is advisable, however, to charge not more than 1,300 lb.

The furnace has replaced most other furnaces since the patents of the inventor expired, proving the general favor it has won. A set of retorting furnaces is arranged in two ways. They are

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either placed on both sides of a horizontal flue in such a way that the openings into it shall not be opposite each other, or they are built, say eight, around a central stack, each flue extending into the stack and then continuing upward for a few feet. With both arrangements any obstruction of the draft is avoided.

2. *The Tatham Furnace* (Figs. 405 to 407).—This is a stationary crucible furnace. The retort, which holds 500 lb. of zinc crust, has the usual inclined position. It rests on a support at the back and protrudes at the front through a small arched

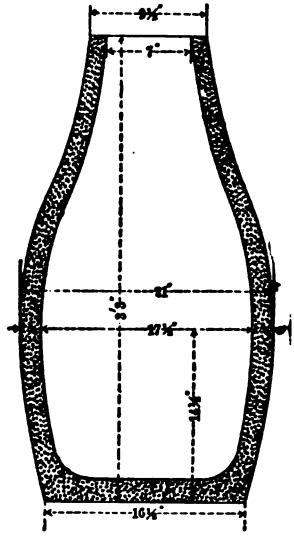
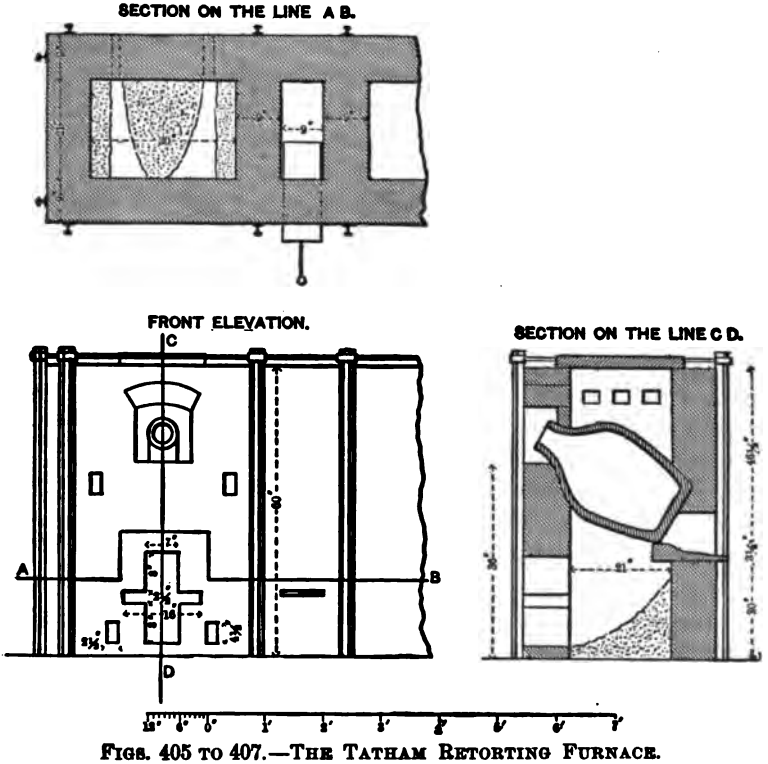


FIG. 404.—GRAPHITE RETORT.

opening. The top of the furnace is covered with a clay tile; the products of combustion pass off through three small openings on one side, leading into a flue that is common to two furnaces, and terminates in the main flue leading to the stack. Each flue can be shut off by a damper. The bottom of the furnace, inclining from the back to the front, is made of brasque. It has a ridge in the center, and is elevated at the sides in order that any lead may run directly through the two gutters toward the front and out of the furnace. The coke is fed at the top, and the ashes and clinkers are removed at the bottom through the large open-

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ing in the front. Both front and back have stoking-holes. The admission of air is regulated by openings which are closed by bricks. The rich lead is tapped from the bottom of the retort, and the residue raked out through the neck. The tap-hole is bored 1½ in. away from the side wall. The entire contents of the retort can, however, be removed through the neck, as is done



FIGS. 405 TO 407.—THE TATHAM RETORTING FURNACE.

with most stationary retorts. The ladle used for this purpose is a 6-in. piece of 3-in. gas-pipe, closed at one end, and having an iron rod 5 ft. long riveted to the other. It is good for six charges. Steitz* several years ago constructed a siphon to discharge the contents of a stationary retort. Sometimes it worked well and again it did not. As no reliance could be placed on it, all attempts at using it have been abandoned.

* Egleston, "Silver, Gold, and Mercury," New York, 1887, vol. i., p. 102.

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This furnace has replaced the Brodie furnace* at the Delaware Lead Works, which had two retorts, one above the other. The results obtained with the Tatham furnace by the writer have been very satisfactory.

3. *Other Furnaces.*—In addition to the furnaces described, which are heated with coke, a few furnaces are in use that are constructed like a reverberatory furnace for the use of bituminous coal. The fireplace may be built on the side of the retort, at the back, or at the front, the aim always being to expose the retort as much as possible to the full action of the flame. The writer has worked a retort fired from the side with good results; retorts fired from the back appear to be satisfactory, while firing from the front has not proved so effective. The reason that retorts heated with bituminous coal have in so many cases given way to those using coke, is because they require so much care to keep the temperature uniformly at a white heat, which is absolutely necessary if the retorting is to be finished in the required time. Further, the facility with which the hot contents of a tilting retort can be discharged has probably influenced the discarding of stationary retorts. The use of gaseous fuel has not been successful here as it has in Germany;† coal oil, however, has in many instances replaced the coke with satisfactory results.

§ 122. CONDENSERS.—The condensers used for collecting the zinc differ very much in form and material. Some are simply old retorts; others are plumbago crucibles (diameter at bottom, 7 in.; at top, 11 in.; height at front, 18 in.; at back, 22 in.; thickness, 1 in.). The former are supported by a specially constructed buggy; the latter rest on tripods, which also hold the receivers for the distilled zinc. Again, they are made of cast iron having the form of a truncated cone. One condenser of this class is 2 ft. long, and has handles on either side by which it is suspended on two hooks from the iron frame of the furnace. A conical condenser is also made of clay; it is about 3 ft. long and supported by a tripod. Another form is that of a sheet-iron cylinder lined with specially molded firebricks. At the base it has, in addition to the tapping-hole, two small pivots to hold a thin chain which is hooked to the frame of the furnace, thus supporting the condenser.

* "Transactions of American Institute of Mining Engineers," iii., p. 324.

† Rösing, *Loc. cit.*

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The condensed zinc is rarely allowed to run off continuously. Usually there is a tap-hole closed by a clay plug or a conical piece of coke, and the condensed zinc is discharged into a suitable mold only three or four hours after the distillation begins. It has been found that more zinc will collect in the condenser if it contains some liquid zinc. Most condensers have a second opening on the upper side for fumes to pass through while the distillation is going on. They go into a sheet-iron pipe leading to the main flue. It has always been considered essential to have this second opening if the distillation is to proceed in a satisfactory way, but at some works using an old retort as condenser the second opening has been dispensed with, the condenser being put in place only when the distillation begins, and not, as is customary, when the retort is well filled.

§ 123. THE METHOD OF WORKING.—The method of working is about the same whichever furnace is used. When a new retort has been put in place, it is carefully warmed and brought up to a dull red heat. It is then ready to be charged. The zinc crust, mixed with from 1 to 3% of charcoal, is brought in an iron wheelbarrow to the front of the retort and charged with a trough-shaped scoop, filled on the wheelbarrow with a small shovel. The temperature is then quickly raised to a white heat; the crust softens and sinks in the retort after half an hour, is pushed down, more crust is added, and this is repeated till the retort is completely filled. The larger rim of the condenser receives a heavy lute of clay, is passed over the neck of the retort, and is made to adhere to the front wall of the furnace; the lower end rests on its support. About an hour after charging, blue powder and then metallic zinc begin to collect in the condenser.

The behavior of tellurium,* which so readily enters the zinc crust in desilverizing, is of interest, since large amounts of telluride gold ores are being treated by lead smelters. In the subjoined table are given some analytical data of the products obtained in retorting tellurium-bearing zinc crusts from Pertulosa, Italy. The three successive zinc taps, with their increasing percentage of tellurium, show that this is not as easily distilled off as the zinc. The fact that the zinc adhering to the iron condenser, and also the retort bullion, retain considerable tellurium, substan-

* Heberlein, *Berg- und Hüttenmännische Zeitung*, 1895, p. 41; "The Mineral Industry," iv., p 480.

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tiates this. Samples of the carmine-colored telluride of zinc adhering to the iron condenser, and to some of the zinc, gave as much as 30% of tellurium, the rest being made up by zinc and the impurities shown in the table. Telluride of zinc changes its color to a grayish black in a few days if exposed to daylight. If it comes in contact with air while hot it is quickly oxidized; the reducing atmosphere in the condenser permits the condensing of the alloy. Some alloy, however, is oxidized, as seen from a selected sample of blue powder which is very rich in tellurium.

Products.	Te. (a)	Se.	As.	Sb.	Sn.	Bi.	Cu.	Cd.	Fe.	Zn.
First zinc tap	0.0080	.....	.....	.....	.....	.....	.....	.....	.....	.....
Second zinc tap	0.0051	.....	.....	.....	.....	.....	.....	.....	.....	.....
Third zinc tap	0.0150	.....	.....	.....	.....	.....	.....	.....	.....	.....
Skin of zinc, lining condenser	0.0870	.....	Trace	Trace	.....	.....	.....	.....	0.2270	.....
Zinc telluride	30.0000	.....	0.0180	0.0230	.....	.....	.....	.....	0.4090	(a)67.0240
Blue powder	3.4600	.....	.....	.....	.....	.....	.....	.....	.....	.....
Retort bullion	0.5156	Trace	1.4095	0.3019	None.	0.0237	2.2477	0.0052	0.2205	0.8537
Gas from condenser	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

Products.	Ni,CO.	S.	Ag.	Au.	Pb.	CO_2. (b)	CO. (b)	N. (b)	C.
First zinc tap	.....	.....	.....	.....	.....	.....	.....	.....	.....
Second zinc tap	.....	.....	.....	.....	.....	.....	.....	.....	.....
Third zinc tap	.....	.....	.....	.....	.....	.....	.....	.....	.....
Skin of zinc, lining condenser	.....	Trace	0.0060	.....	1.4700	.....	.....	.....	Trace
Zinc telluride	Trace.	.....	0.0040	.....	2.5230	.....	.....	.....	Trace
Blue powder	.....	.....	.....	.....	.....	.....	.....	.....	.....
Retort bullion	0.0068	Trace	5.4616	0.0564	(a)88.8974	.....	.....	.....	.....
Gas from condenser	.....	.....	.....	.....	.....	1.4	55.2	43.4	.....

(a) By difference. (b) Volumes.

The distillation is finished after from five to six hours, according to the size of the charge, the percentage of zinc in the crust, and the draft of the furnace. Too much emphasis cannot be laid upon the draft. The slightest obstruction means a failure in distilling off the zinc in the required time, and many consequent disturbances, inconveniences, and losses. The main points to be looked after during the operation are the quick raising of the temperature, and the keeping it high. The higher the temperature, the more rapid will be the distillation, the better the output of metallic zinc, and the lower the percentage of zinc remaining in the rich lead. If the temperature is lowered, blue powder forms, and some of the crust floating on the lead may harden and be suddenly broken by the zinc vapors developed beneath the crust, when the heat is raised again, and thus cause

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an explosion. The effect of this is to loosen the condenser from the neck of the retort. At most works it is the practice to introduce at certain intervals, through the upper opening in the condenser, an iron rod, free the neck of the retort with it from oxidized zinc, and then stir up the crust floating on the lead. But the saying that the condenser ought to smoke well if the distillation is to go on satisfactorily, is a mistaken one because a condenser with only one opening—the tap-hole, which is kept closed—gives excellent results.

When the zinc ceases to collect in the condenser the distillation is finished. The last zinc is then removed and the condenser taken off and scraped clean. In the meantime the vapors in the retort pass off into the air. This is assisted by throwing a few chips of wood into the retort, which has now to be emptied. With a tilting furnace, a slag-pot, lined with brick to prevent the hot metal from cracking the cold bottom, is wheeled in front of the furnace and the lead emptied into it. While the lead is running out, the sample for assay is taken. This is then allowed to cool and, after the removal of the dross from the surface, is ladled into small molds, so as to obtain bars of suitable size for the subsequent treatment in the English cupelling furnace. The residue in the retort, consisting of slag and charcoal, is removed with an iron scraper. It is essential for the life of the retort that it be well cleaned after each distillation. The retort is now raised, and some fine charcoal thrown into it to prevent the oxidation of the globules of lead adhering to the sides; it is then turned back to its normal position, and is ready to be charged again. The grate bars are now cleaned, clinkers adhering to the furnace walls or sides of the retort are removed, fresh coke is added, and all is then ready for the next operation. The time required for discharging and refilling is about 20 minutes. In scraping, care must be taken to do it gently, so as not to wear off the lower side of the retort. To avoid this, as well as to change the line of contact with the surface of the lead, the position of the retort is sometimes changed, after it has worked about 25 charges, by turning it 180°.

With the Tatham furnace the operations are similar, except that the lead is tapped from the bottom, and only the residue raked out through the neck.

§ 124. Tools.—The tools required by one man in retorting are:

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two scrapers (6 ft. long, $\frac{5}{8}$-in. round iron flattened out at one end to the width of 3 in. and bent up 3 in.) to stir the contents of the retort after distilling and to rake out the residue after tilting; two pokers (5 ft. long, of $\frac{7}{8}$-in. iron) to remove the clinkers from the grate; one bar (8 ft. long, $\frac{3}{4}$-in. steel) to break off clinkers from the walls; one scoop to feed the coke; one ladle (6 in. in diameter and 3 in. deep, with a 4-ft. handle); two slag-pots lined with brick; a wheelbarrow to receive the zinc crust; one scoop and shovel to charge the retort; two molds for each retort to receive the zinc; 10 bullion molds.

§ 125. Results.—The weight of the charge is from 1,000 to 1,300 lb. of zinc crust, and it takes from five to six hours for one operation; less time is required in winter than in summer. The crust yields from 70 to 80% of lead containing from 4 to 10% of silver and from 0.75 to 1.50% of zinc. The zinc recovered in the form of metal is about 10% of the weight of the crust, and that in the form of blue powder about 1%. Of all the zinc required for desilverizing over 60% is recovered to be used again in the kettles. This has been raised to 70% with the Howard stirrer and press. The quantity of dross varies a great deal; from 5 to 8% of the weight of the crust was a fair figure, before the Howard improvements. A retort lasts now 40 charges; formerly 25 charges was considered a good average. For every ton of crust 1,100 lb. of coke are required. To use a good quality of coke of uniform size (egg coke), although more expensive at first, is cheaper in the end, when the life of the retort and the better results obtained are considered. One man attends to from two to four furnaces in a 12-hour shift.

As there is liable to be some leakage in a retort, which escapes attention, it is advisable not to throw away the coke ash, but to add it to the blast-furnace charge. Fluedust from retort ovens showed Zn 12.2%, Ag 16.8, and Au 0.20 oz. per ton (Iles). As stated above, reduced oil has, whenever the cost makes it possible, replaced coke or coal. With a 1,000-lb. charge the 550 lb. coke in six hours are replaced by 52 gal. oil in six hours with steam injection, and 26 gal. in five hours with air injection; in both cases the Reid burner was placed at the side.

Roesing* proposes to facilitate the distilling of zinc crust in a very novel way. He heats the retort, having a basic lining in a

* *Berg- und Hüttenmännische Zeitung*, 1890, p. 369.

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tilting furnace. When the crust has become soft, he introduces some fuel to drive out all the air, and pours pig-iron, heated above its melting point, into the retort, which volatilizes the zinc to be caught in a condenser. By tilting the retort, first the iron is poured off to be used over again and then the rich lead.

§ 126. Comparison of the Two Processes.—In comparing the smelting of the zinc crust in the blast furnace with distillation, it is clear that if the cost of distilling is made up by the amount of zinc recovered, it is preferable to smelting, on account of the greater loss of silver incurred in the latter method. General practice has shown that there is even a margin of profit in distilling the silver crust, and hence the smelting has been almost entirely abolished. There is one case when it may be doubtful which is the more profitable. The gold crust is very heavy, compared with its low contents in silver. In retorting, very little metallic zinc is recovered, most of it being obtained in the form of blue powder; the retort bullion assays from 400 to 800 oz. of silver per ton and is rich in copper. Some works, therefore, prefer to smelt this crust with the addition of matte, obtaining thus a bullion which is free from copper and is easily cupelled. Since the base bullion produced in blast furnaces has become so rich in gold on account of the large amounts of telluride ores treated, few, if any, refiners make a separate gold crust.

§ 127. Treatment of By-Products.—This is a very important part of refining. It requires to be carried on simultaneously with the main operations, so that the by-products may be disposed of at once and not allowed to accumulate.

§ 128. Softening Furnace Dross and Skimmings.—According to one method of working, these products are first liquated in a reverberatory furnace, in order to extract as much lead and silver as possible. One furnace of reasonable dimensions, if kept constantly in use, is able to liquate both products of a plant treating 100 tons base bullion per day. While a furnace for liquating the dross ought properly to have a somewhat different construction from the one used to treat the antimony skimmings, it is found more economical to have one furnace for the two operations. The reverberatory (for example, one with a hearth 8 by 12 ft. and 10 in. deep, and having two working doors on one side only) is built just like a softening furnace. It also has a water jacket to counteract the corroding influence of the anti-

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mony skimmings when they are being melted down. The two tap-holes of such a furnace are placed in one of the sides near the center.

In liquating *dross*, the furnace is charged and heated until the lead that has liquated forms a bath on which the dry dross floats, the operation being continued until sufficient dry dross has accumulated on the lead, to be raked out through the furnace door. The dross directly in contact with the lead is not touched till toward the end. When the level of the lead comes near the furnace door, enough is tapped into an outside kettle to leave a bath of lead for the charge next following to float on; charging, heating, and drossing are continued until the batch of dross is worked up. The temperature is always kept low, in order that no dross may be taken up again by the lead. The liquated dross is smelted in the blast furnace with a sulphur-bearing ore or by-product (galena, matte low in copper) to form a matte, or to raise the grade of the matte that has been added to the charge. The lead goes to the softening furnace.

In liquating the *softening furnace skimmings*, two objects are aimed at, the elimination of metallic lead and the desilverization of the skimmings. Both are accomplished at once by melting down the skimmings with a reducing (*i.e.*, a smoky) flame in the reverberatory furnace. All the lead carried off from the softening furnace will collect on the bottom of the hearth; from the melted skimmings floating on top, part of the lead oxide, converted into metal by the reducing flame, will, while descending through the skimmings, desilverize these and carry the silver with it into the metal bath below. If necessary, some fine coal is spread over the charge. When the furnace is filled and everything well melted, the liquated skimmings are tapped from the upper tap-hole into slag-pots, and the lead from the lower tap-hole into a kettle, whence it is ladled into bars to go back to the softening furnace. Another and better way is to have only one tap-hole, and to tap the entire contents of the furnace into a kettle that has an overflow spout. While the lead is flowing into the kettle some can be ladled out, if necessary. As soon as the liquated skimmings appear, they will rise in the kettle and overflow through the spout into slag-pots placed beneath. When the flow ceases, the skimmings still floating on the lead will soon

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harden, and are then removed; the lead in the kettle is cast into bars and goes to the softening furnace. When the cake of liquated skimmings is rolled out from the slag-pot and broken, there will be found two products: liquated skimmings assaying 3 oz. silver or less to the ton, forming the bulk of the cone, and at the bottom a small cake of a silvery-white antimonial speise of lead and copper, which contains as much as 40% copper, and, with 250-oz. bullion, assays often as high as 300 oz. silver to the ton. Formerly it was considered preferable to tap the skimmings or let them overflow on an iron plate, as they chilled quickly and were easily broken to pieces, but they ran higher in silver.

The liquated skimmings are smelted in a small (36-in.) blast furnace, with about 100% of slag and 11% of fuel, care being taken to avoid a slag that is very ferruginous, as speise might otherwise form. A slag frequently made is SiO_2 30%, FeO 50%, CaO 10%. A speise produced in smelting skimmings contained, according to two analyses made by F. H. Davis in 1896, from a sample furnished the writer by A. W. Jenks:

Pb.	Sb.	Cu.	Fe.	Zn.	S.
Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
79.96	16.62	0.327	0.157	0.271	0.026
78.67	18.22	0.860	0.142	0.251	0.026

Another sample gave Pb 63.3% and Sb 30.5%, the antimonial lead produced assaying 18% antimony. The speise has a beautiful amethystine color, shows cleavage planes similar to those of spiegeleisen and is sectile. In order to reduce the loss of antimony by volatilization (which is considerable), non-argentiferous galena is added to the charge. The sulphur acts as a reducing agent, doing thus to some extent the work of carbon or carbon monoxide. The amount of galena added is so regulated that no matte shall form; it is from 13 to 28% of the weight of the skimmings. While this addition reduces the percentage of antimony in the resulting hard lead, this is of no consequence, as, if hard lead assays from 14 to 20% antimony (a common figure), this is paid for at the same rate as when the lead contains from 7 to 10%. In order to reduce the copper contained in the skimmings, as much as 25 lb. of gypsum is added to the charge. The resulting

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slag is liable to be rich in lead, although this is not always the case, and to contain considerable antimony. Smelting the slag again in the ore blast furnace would bring the antimony back to the base bullion, and with every smelting of hard lead fresh ore slag would be required. Therefore, a contain amount of slag is often kept apart and run over and over again, and only the small excess, produced in every run from coke-ash and fluxes, given back to the ore-charge; the rest is saved for the next antimony run.

Another way to treat the softening skimmings is to smelt them, without liquating, in the blast furnace and to make a hard bullion. This is charged into the softening furnace with soft lead poor in silver (say from the Mississippi Valley), in the proportion of one hard lead to four soft lead. The amount of silver in the original hard bullion is reduced, the softening facilitated, and the resulting skimmings are low enough in silver to be smelted directly into marketable hard lead, which, however, ordinarily runs as high as 6 oz. silver per ton.

The grade and grain of the hard lead are much improved by poling it in a kettle for a few hours or liquating it in a reverberatory furnace at a low temperature. The reason for this is that the copper remaining in the softening skimmings enters the hard lead on being smelted. The dross from liquating the crude hard lead has been found to contain as much as 40% copper, but generally contains about 10%. The hard lead is usually molded by ladling from the kettle where it has been poled, or from the one into which it ran from the liquating furnace. The surface of the refined hard lead, when liquid, soon becomes covered with dross, and in order to obtain clean bars without being forced to waste much lead by skimming, it is advisable to place a wrought-iron ring on the lead just large enough for the ladle to pass through and to dip the lead from the ring. Thus only a very small surface will have to be kept bright.

The latest way of treating the dross and skimmings is to smelt them just as they come from the softening furnaces in a reverberatory furnace with some galena free from silver. The result is base bullion, a copper matte free from iron, and refined skimmings free from copper. A furnace made for this purpose is built into an iron pan, 10 ft. 6 in. long by 9 ft. 6 in. and inclosed by water jackets. It has a lead well; the fire-box is 4 by 4½ ft.;

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a Körting injector furnishes the undergrate blast. The charges, weighing 635 lb. (350 lb. dross, 150 skimmings, 100 galena, 20 sand, 15 fine coke), are dropped from the hopper in the roof in quick succession until the hearth is filled, and replenished as fast as they melt down, so that the fusion takes place on a bath of lead. The surplus of lead is ladled out from the well; speise, matte and skimmings are tapped into an overflow slag-pot. In 24 hours 90 charges are worked, yielding 16,000 lb. of base bullion, a small amount of speise (Cu 50%, Pb 17%, Ag 117 oz., Au 2½ oz.), 3,500 lb. matte (Cu 35 to 40%, Pb 30%, Ag 100 oz., Au 0.3 oz.), and 5,000 lb. clean skimmings (Pb 75%, Ag 1 to 1½ oz.). The reason that such a large amount of material can be smelted in a comparatively small furnace is that it is treated on a bath of lead, and that the furnace never becomes cool. The skimmings are low enough in silver to be smelted for hard lead, which being free from copper and other impurities is ready for market when ladled from the lead-well of the blast furnace. Thus the common operation of liquating or poling hard lead from the blast furnace is made unnecessary. The bullion goes back to the softening furnace. The matte is converted in a cupelling furnace with some silicious material into copper bottoms, 60% copper matte and slag, the bottoms collecting the gold. The furnace is similar to the one represented in Figs. 285 to 289. The test ring, which is not water-jacketed, is 6 by 8 ft., and is lined with a mixture of raw and burned clay; the cavity is 6 in. deep. The slag runs off from the front into a iron pot and, when the test is full of metal and matte, the contents are tapped into molds, placed on a truck beneath the test, by drilling with an augur a hole into the breast. The tap-hole is again closed from the inside by inserting a clay stopper at the back through the tuyere-hole; the furnace is then ready for another charge. When the molds are cold the bottoms are separated from the matte, and this is converted in another cupelling furnace into metallic copper to be used in precipitating silver from silver sulphate solution obtained in parting doré silver by means of sulphuric acid. For every 100 lb. metallic copper produced, 26 gal. of reduced oil are required, an air-injector being used. The advantage for refining works in concentrating the copper obtained from base bullion in this way lies in the fact that copper refiners pay for only 95% of the silver contained

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in the matte, make no allowance for the lead, and charge two and a half times as much for desilverizing matte as they do for metallic copper.

ANALYSES OF HARD LEAD AND HARD LEAD DROSS.

	Lautenthal.		Clausthal.	Pribram. (d)	Pueblo Smelting and Refining Co. (e)
	Before Poling. (a)	After Poling. (b)	Dross from Liquating Hard Lead. (c)		
Sb	12.753	15.390	38.763	24.270	20.550
As	.....	.....	.....	0.109	0.100
Cu	1.861	0.152	37.643	0.160	0.300
Pb	85.291	84.650	23.962	74.886	79.042
Ag	0.0085	0.008	.....	0.009	0.008
Ni	.....	.....	(S 0.240)	0.015	.....
Fe	Trace.	Trace.	0.139	0.018	.....
Zn	Trace.	Trace.	0.232	0.009	.....
Sn	.....	.....	.....	0.594	.....

(a) Private notes. (b) *Ibid.* (c) *Berg- und Hüttenmännische Zeitung*, 1870, p. 169. (d) *Oesterreichisches Jahrbuch*, xxxix, p. 64. (e) Howard, Private notes, 1886.

§ 129. Tin Skimmings.—Tin ore occurs so rarely with lead-silver ores that it is only an exceptional case when tin skimmings are formed in softening base bullion. At Freiberg (Saxony) the tin of the ores becomes concentrated to some extent in the base bullion produced. The skimmings that rise to the surface in the softening furnace after the dross has been removed have the following composition:* PbO, 70.35%; SnO_2, 12.53%; Sb_2O_5, 12.50%; As_2O_5, 4.73%; CuO, 0.61%; and contain 72.9 oz. silver to the ton. Plattner has introduced a method of desilverizing these skimmings and concentrating the tin in a marketable alloy which contains Sn, 33%; Sb, 14%; As, 1%; the rest being lead. Details with analyses of intermediary products are given in the reference.

§ 130. Kettle Dross.—The impurities contained in this dross are very few. It consists principally of lead oxides mixed with metallic lead, and is usually put back into the softening furnace with the next charge after the furnace dross has been drawn off. This is the best way of disposing of it.

§ 131. Refining Skimmings and Polings.—An analysis of refining skimmings gave Iles, SiO_2, 1.6%; Fe, 0.6%; Zn, 8.0%; Cu, trace; As, none Sb, 4.02%; Pb, 75.0%; Ag, 1 oz.; and Au, trace. The percentage of antimony is exceptionally high. It is also

* Plattner, *Berg- und Hüttenmännische Zeitung*, 1863, p. 417.

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singular that there should be present traces of copper and gold. The skimmings are best treated in a reverberatory furnace that is used for no other purpose, except perhaps for liquating hard lead, in which case the lead resulting from the refining skimmings is sold as second-class lead on account of the copper it has taken up. If the reverberatory is used only for reducing refining skimmings, then resulting lead can be worked in with the regular refining charges and corroding lead produced. The hearth of a furnace suited for this purpose may be 10 by 5 ft., and 9 in. deep, built of firebrick inclosed in a wrought-iron pan. It will have an inclination of 3 in. from the bridge to the flue, where the main working door is placed; a second door is in the middle of one of the sides to introduce and distribute the charge. The tap-hole below the flue discharges the lead into a small spherical kettle having a fireplace beneath. The English (§ 44) or Silesian (§ 45) reverberatory furnace also serves the purpose very well.

The mode of operation is as follows: The hearth of the furnace, dark-red from a previous charge, is covered with a 2-in. layer of fine coal to protect it from the corroding effect of the skimmings. It is then filled with skimmings mixed with 10% of fine coal, leaving room for the gases to pass. Sometimes the charge reaches only to the working doors, being renewed from time to time as it shrinks. As the charge heats, lead will flow into the sump, whence it is tapped at intervals. The fire is gradually urged; when no more lead flows the charge is rabbled, and when exhausted drawn.

Four tons of skimmings, yielding about 60% lead, are worked in 12 hours, two men attending the furnace.

The residue, which contains lead, zinc, antimony, perhaps some arsenic, and coal-ashes, is added to the smelting charge of liquated softening skimmings.

The refining skimmings and polings are sometimes, though rarely, smelted in the ore blast furnace with the original softening skimmings to reduce the silver contents of the latter. They are also sometimes charged into the softening furnace after the dross has been removed, in order to oxidize the arsenic and antimony of the bullion, but some zinc enters the hard lead when the skimmings come to be smelted in the blast furnace, making it unfit for some of its uses, as for example in acid works.

Owing to the scarcity of lead ores now prevailing, nearly all

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refiners use up their refining skimmings for lead-flux in smelting dry argentiferous ores.

The working-up of oxides obtained in refining in the kettle has already been referred to. It may be added that in order to sell this product readily refiners have in some instances been forced to grind it up with oil. The pigment has a good body and forms an excellent material for ground work.

§ 132. Rich Lead and Metallic Zinc.—The former goes to the cupelling furnace (§ 139); the latter is used in desilverizing. It always contains a few ounces of silver.

§ 133. Retort Dross and Blue Powder.—The retort dross is very rich in silver, which must be extracted quickly. The dross, if formed in small quantities, is worked off in the regular cupelling a little at a time, or at some works it is charged back into the retort. If there is too much for this, it is allowed to accumulate, and is scorified on the bath of lead free from or low in silver, with which a new test of the cupelling furnace is usually charged. Sometimes the retort dross is added to the softening furnace, after its own dross has been removed, that the silver may be taken up by the lead, and the impurities oxidized and taken up by the skimmings. It is not often that the dross is added to the charge of the bullion blast furnace.

The blue powder, consisting of a mixture of finely divided metallic zinc and zinc oxide, always contains some silver, say from 4 to 5 oz. per ton. It is not readily disposed of. If distilled by itself with the addition of charcoal, it yields from 33 to 55% of zinc. At some works it is fed back to the retort with the following charge of zinc crust; at others it is added with the first zinc to the desilverizing kettle, that the metallic zinc may be taken up. If this is done, 50 lb. of blue powder are charged at a time into the kettle before the lead is tapped into it from the softening furnace. It is stirred into the bullion while this is rising in the kettle. It does not remove much silver from the kettle, but serves rather to saturate the lead with zinc and to assist in removing the gold and copper. Sometimes the blue powder is screened to remove all coarse particles and is then sold to zinc works. In some European works it is dissolved in sulphuric acid and the zinc vitriol sold or converted into Lithopon (§ 67). Happily, since the use of the Howard stirrer and press, these products are

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being formed in small quantities only and have lost the importance they formerly had.

§ 134. LITHARGE is reduced in a reverberatory furnace or goes to the bullion blast furnace; sometimes it is added to the softening furnace after drossing. While the reduction of litharge in the reverberatory furnace is preferable from a purely metallurgical point of view, the blast furnace is in more common use, as refining works are thus enabled to smelt dry silver ores at a smaller charge than Western smelters, who suffer from scarcity of lead ores. It is important for a refinery to smelt some ore in treating by-products in order to make new slag, and not be forced to smelt old slag over and over, which is expensive and causes losses in lead and silver, as these slags become gradually rich in zinc.

§ 135. OLD RETORTS, CUPEL BOTTOMS, ETC., are added to the charges of the bullion blast furnace.

§ 136. TABLE OF DESILVERIZATION.—The following table gives a summary of the work done in an American desilverizing works, as shown in detail in the preceding pages:

1. Base Bullion: softened in reverberatory furnace yields soft bullion (2), softening dross (7), softening skimmings (8).

2. Soft Bullion: desilverized in kettle yields kettle dross (back to 1), gold crust (3), silver crust (11), desilverized lead (12).

3. Gold Crust: liquated yields liquated gold crust (4), and gold-bearing base bullion (2.)

4. Liquated Gold Crust: retorted yields retort dross (15), blue powder (14), zinc (2), retort (doré) bullion (5).

5. Retort (doré) Bullion: cupelled yields doré silver (6), litharge (17), cupel-bottom (17).

6. Doré Silver: sold, or parted by sulphuric acid or electrolysis, yields gold, silver, blue or green vitriol, or precipitated copper.

7. Softening Dross: liquated in reverberatory I. yields base bullion (1), and liquated dross (17), or smelted with (8).

8. Softening Skimmings: liquated in reverberatory I. yields base bullion (1), and liquated skimmings (9), or smelted with 7 and galena in reverberatory III. yields base bullion (2), matte (16), copper bottoms (sold), and skimmings (9).

9. Liquated Skimmings: smelted in 18 yield hard lead (10), and slag (18).

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10. Hard Lead: poled in kettle or liquated in reverberatory II. yields market hard lead and dross (16).

11. Silver Crust: goes to 3, 4, and 5 in separate apparatus, yielding fine silver and by-products treated like 4 and 5.

12. Desilverized Lead: refined in reverberatory yields corroding lead and refining skimmings (13.)

13. Refining Skimmings: reduced in reverberatory II. yield lead (either to 12 or sold) and dross (18), or smelted with ore in (16.)

14. Retort Dross: either fed on cupel (5), or goes back to retort (4), to softening furnace (1), or to bullion blast furnace (17).

15. Blue Powder: goes either to 2 (as first zinc), or back to retort (4), or is screened and sold.

16. Matte: smelted in reverberatory similar to cupelling furnace, yields metallic copper (sold) and slag (17).

17. Bullion Blast Furnace.

18. Hard-Lead Blast Furnace.

§ 137. GENERAL REMARKS.—A few general remarks regarding Parkes' process are in place.

The output of silver with average base bullion as it comes from Western smelting works (running from 150 to 300 oz. silver, and 0.5 oz. more or less, gold per ton, being pretty free from dross, but containing some arsenic and antimony) is never less than 99.5%, and generally there is a slight surplus of silver; with gold it is from 98 to 100% (a surplus is rare unless the base bullion is very rich in gold); with lead from 99 to 99.5%.

A refinery makes its deduction for loss in treatment and for cost of working, the base bullion being delivered f. o. b. refinery. Some refining works sum up both items and make a general charge by the ounce of silver; *e.g.*, one cent for every ounce of silver, the lead being paid on New York quotations and the freight to New York being deducted. Others make a working charge by the ton and a deduction for loss in silver; *e.g.*, the working charge varies from $10 to $14 per ton, and the reduction amounts to 2 or 3 oz. silver per ton, the lead being paid as above. A number of other ways of settling between smelter and refiner are in use, but the two quoted are the most common ones.

Below is reproduced the scheme of the Kettle-Book, the most important one of all the books. It presupposes that the Howard

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press is used for liquating and shows the various products and the amount of fuel consumed:

KETTLE-BOOK.*

189— Date.		Lot.	Name of Bullion.	Charge.	Gross Weight.	Assay: Oz. per Ton.		Contents: Total Ounces.	
Month.	Day.	No.		No.	Lb.	Ag.	Au.	Ag.	Au.

Softening Dross.	Softening Skimmings.	Kettle Dross.	Net Weight in Kettle.	Assay: Oz. per Ton.		Contents: Total Ounces.	
Lb.	Lb.	Lb.	Lb.	Ag.	Au.	Ag.	Au.

Zinc Added: Lb. for		Gold Crust.	Assay. Oz. per Ton.	Contents: Total Ounces.
Au.	Ag.	Lb.	Ag.	Ag.

Silver Crust.	Refining Skimmings.	Refined Lead.	Coal.	Remarks.
Lb.	Lb.	Lb.	Tons.	

* Owing to the form of this volume, it has been necessary to put the headings in four rows one beneath the other, while in the actual kettle-book they simply run across the two opposite pages in one line.

No detailed statement can be made about the cost of refining that would be generally applicable, as the single items vary greatly in different refineries. In a general way it may be said that the actual operating expenses, including interest and taxes, vary from $5 to $6 per ton of base bullion. If the general ex-

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penses, such as salaries, marketing, etc., be added, this figure rises to $7 and $8. If, finally, the loss in metal and incidentals be added to the last figure, giving the total final cost, this will be found to vary from $10 to $12 per ton.

In a refinery where so many by-products result, it is essential to know how much silver, gold, and lead is contained in each of them. These quantities vary a great deal with the different kinds of bullion treated. It is therefore essential to be able to follow up the metals from the time they enter the refinery until they leave it.

It is to be noted that the "net weight in kettle," before adding the first zinc, is found by deducting the sum of weights of softening dross, skimmings, and kettle dross from the base bullion charged into the softening furnace. The "net weight in kettle" after adding the first zinc, is found in a similar way, by adding the pounds of zinc used for extracting the gold to the net weight before the first zinc addition was made and deducting the weight of the gold crust from this sum.

The other principal books kept in a refinery are: the retort-book, the cupellation-book, and the books for reverberatory furnaces I. (liquating softening-dross and skimmings), II. (reducing refining skimmings and liquating hard lead), and III. (smelting dross and skimmings with galena), and special assay-books for the softening furnace, desilverizing kettle, retorts, and cupelling furnaces.

§ 138. Relative Advantages of Parkes' and Pattinson's Processes.—The many advantages* (§ 97) that the Parkes process has over that of Pattinson have made it *the* desilverizing process used in the United States, there being only one refinery (at Eureka, Nevada) that desilverizes with the Luce-Rozan process. There is, however, one instance in which Pattinson is to be preferred to Parkes. It is when the base bullion is rich in bismuth. During the crystallization bismuth follows the liquid lead (§ 98). It is thus concentrated in the rich lead, and can be recovered when this is being cupelled. The bismuth contents of the market lead thus becomes very much lowered, although not quite removed (§ 98). In Parkes' process the bismuth enters only to a small extent (§ 101) into the zinc crust, with the result that the refined lead may become richer in bismuth than the original bul-

* Phillips, *Engineering and Mining Journal*, May 21, 1887.

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lion. This difference is shown by the two following analyses by Hampe* from Lautenthal market lead where Parkes' process replaced that of Pattinson.

Lautenthal.	Pb.	Cu.	Sb.	As.	Bi.	Ag.	Fe.	Zn.	Ni.
Pattinson.....	99.966200	0.015000	1.010000	None.	0.000600	0.002200	0.004000	0.001000	1.001000
Parkes.....	99.983139	0.001413	0.005698	None.	0.005487	0.000460	0.002289	0.000834	0.000680

An interesting combination of the two processes is found at Freiberg,† where the original base bullion, containing from 116 to 233 oz. silver per ton, and from 0.02 to 0.06, rarely 0.16% bismuth, is concentrated by crystallization to a liquid lead with 0.17% bismuth, assaying 583 oz. silver per ton, which is cupelled, while the crystals averaging 30 oz. silver per ton are desilverized with zinc, the bismuth in the market lead not exceeding 0.02%.

Large shipments of base bullion with considerable quantities of bismuth are of rare occurrence in refining works. Occasional ones are worked in with other bullion that is free from, or at least low in, bismuth.

* *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xviii., p. 195.

† *Engineering and Mining Journal*, Dec. 4, 1886; *Berg- und Hüttenmännische Zeitung*, 1887, pp. 45, 192; 1889, p. 116.

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CHAPTER XI.

CUPELLATION.

§ 139. Introductory Remarks.—The process of cupellation has for its object the final separation of lead and silver. It consists in melting and heating in a reverberatory furnace argentiferous lead with access of air to the temperature at which litharge forms on its surface. This is run off and is in part absorbed by the hearth, while the silver, having scarcely any affinity for oxygen, remains behind in the metallic state. The oxidation of the lead is principally effected through the action of the blast playing over its surface, but is also assisted by the litharge when formed, as this absorbs oxygen and gives it off again to the lead and its impurities. The most important of these are copper, arsenic, antimony, bismuth, tellurium, silver, and gold.

The bulk of the copper is removed with the dross and all the arsenic and antimony with the skimmings, just as in softening base bullion (§ 104). The copper remaining with the lead after drossing, is taken up only very gradually by the litharge. As it has less affinity for oxygen than lead, its oxidation must be caused not so much by the action of the air as by a large quantity of litharge acting on a small amount of copper. With reversed conditions cuprous oxide* oxidizes lead. The oxidizing action of cuprous oxide contained in the litharge seems to be the cause of the fact that cupriferous lead is cupelled more quickly and with less loss in lead and silver than if the lead were free from copper. Kerl† states that in cupelling 30 tons of argentiferous lead containing from $\frac{3}{4}$ to 1% copper, the operation lasted 24 hours less than when copper was absent, and that the losses in metal were as 2 : 7.

* Berthier, "Traité des essais," Liége, 1847, vol. ii., p. 572.
† "Grundriss der Metallhüttenkunde," 1881, p. 270.

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Bismuth is concentrated with the silver in the lead until toward the end of the process, and then greatly retards the progress of the work. It is finally oxidized and enters the litharge, giving this a greenish color, and is absorbed by the hearth material, while the silver also retains some bismuth with great tenacity. If the bismuth is to be recovered, the silver is concentrated only to a certain degree (say to 50 or 60%) in the lead, which is then cupelled in a separate furnace; the saturated part of the hearth of this furnace and the litharge form the raw material for the extraction of bismuth in the wet way.

Tellurium* has a strong affinity for silver, similar to that of copper. In cupelling retort bullion with 0.31% tellurium at Pertulosa, Italy, the following products were obtained:

Products.	Te.	Se.	As.	Sb.	Pb.	Ag.	Au.
First litharge	0.2170	.....	.....	.....	.....	.....	.....
Crude silver while brightening.	0.0487	Trace.	Trace.	0.0197	0.1785	(a)96.2711	0.6190
Second litharge	3.3437	Trace.	Trace.	0.0972	74.3440	1.1292	0.0008
Refined silver	0.0130	Trace.	Tracc.	Trace.	0.0088	98.4937	0.6340

Products.	Cu.	Bi.	Fe.	Zn.	Ni,Co.	Cupel Bottom.	O and Loss.
First litharge	.....	.....	.....	.....	.....	.....	.....
Crude silver, while brightening.	1.9332	0.0140	0.0113	0.0032	0.0013	.....	.....
Second litharge	5.9709	0.0538	0.0608	0.0211	0.0041	2.4580	12.5174
Refined silver	0.8454	0.0028	0.0073	Trace.	Trace.	.....	.....

(a) By difference.

A sample of the silver shown in the table, taken during brightening, had a bluish-gray color, was brittle and showed a fine-grained fracture. In order to remove the impurities, some lead was added, giving the second litharge and refined silver. (See § 150.) Of the tellurium contained in the crude silver, 61% entered the litharge, 1.4% remained in the refined silver, and the difference, 37.6%, was volatilized or absorbed by the hearth material. The volatile character of tellurium was clearly shown by assays of fluedust, samples from 70 to 90 ft. away from the furnace being richer in tellurium than those close by.

Silver in the litharge, is probably always present both as oxide and as metal. Rose† states that silver oxide begins to

* Heberlein, *Berg- und Hüttenmännische Zeitung*, 1895, p. 41; "The Mineral Industry," iv., p. 481.

† "Handbuch der Analytischen Chemie," Leipsic, 1867, vol. i., p. 339.

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lose its oxygen at 250° C; according to Sainte Claire-Deville and Debray,* as well as Troost and Haute-Feuille,† the oxide appears to be able to exist at a very elevated temperature. Wait‡ studied the conditions under which silver is oxidized at high temperatures and found that on heating finely divided silver with metallic oxides the formation of silver oxide varied from a trace to 39%, according to the oxide used, the temperature and the length of time it was exposed to heat. He§ also dissolved from litharge containing 2.94% silver, by means of acetic acid, 18.67 and 19.25% of the silver present. As metallic silver is insoluble in acetic acid, the silver dissolved must have been present as oxide. If silver-bearing litharge remains in contact any length of time with metallic lead having little silver, it loses its silver. Thus at the beginning of the cupellation little silver is contained in the litharge. During the progress of the operation the lead becomes richer, more silver is liable to be oxidized, and less of it again reduced by the enriched metallic lead. Fine particles of argentiferous lead cannot be prevented from being carried away by the litharge. From both causes, therefore, the tenor in silver of the litharge will increase with the progress of the cupellation.

Gold, finally, follows the silver in the cupellation, but none of it, or perhaps a trace, is found in the litharge.

According to the general construction of the furnace, and the consequent mode of operating, cupellation is generally discussed under two heads: A. German Cupellation;‖ B. English Cupellation.

A. German Cupellation.

§ 140. Characteristics.—The characteristics of this method are: a large reverberatory furnace with a fixed bed and a movable roof, that the bullion to be cupelled is all charged at once and that the silver is not refined in the same furnace where the cupellation was carried on.

* Graham-Otto-Michaelis, "Anorganische Chemie," Brunswick, 1884, iii., p. 985.

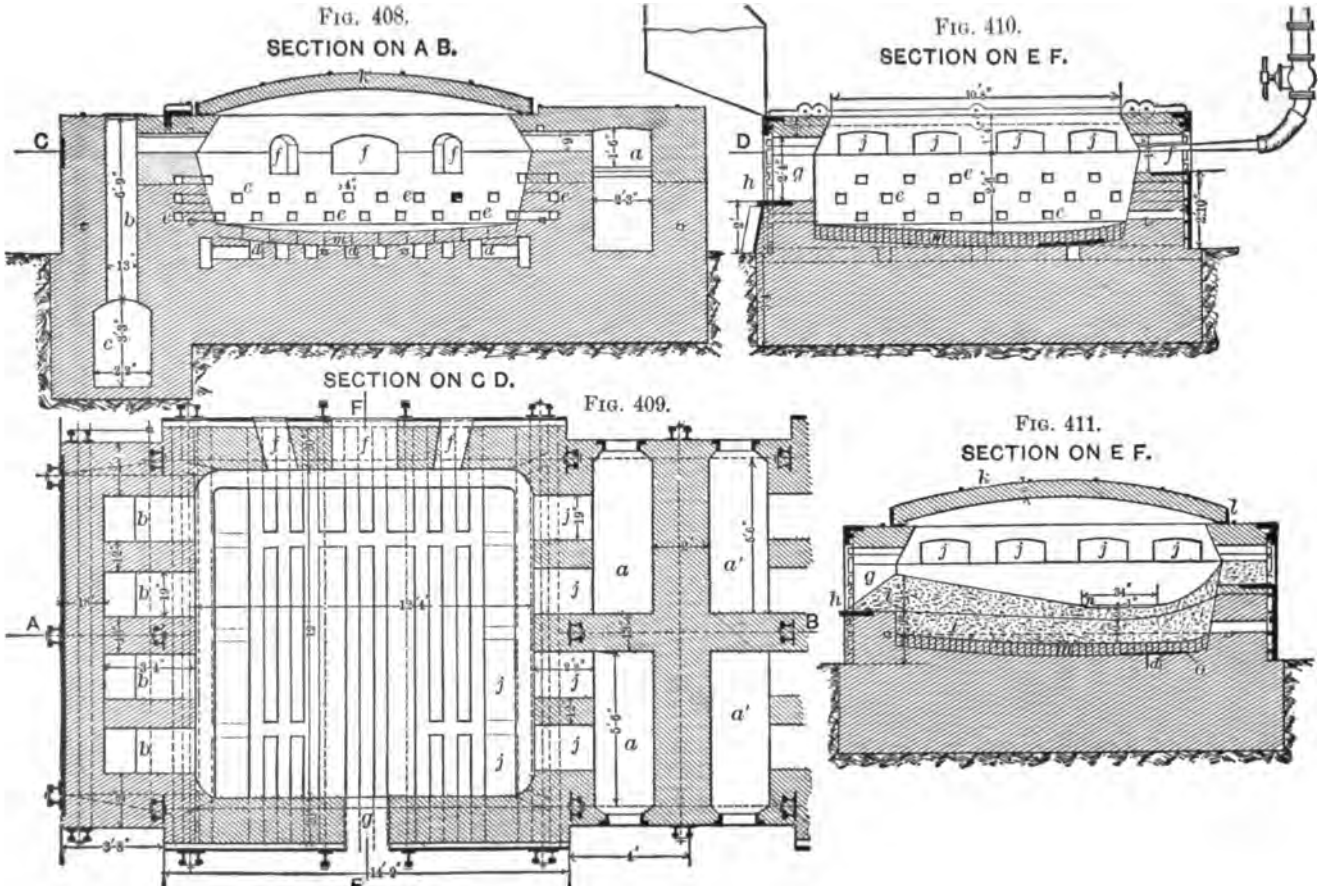
† *Ibid.*

‡ *Journal of the American Chemical Society*, xviii., p. 254.

§ "Transactions of American Institute of Mining Engineers," xv., p. 468.

‖ The writer is fully aware that the German cupelling furnace is not much found in this country; in fact, he knows of only one refinery that uses it. If, nevertheless, it is treated here in more detail than may seem necessary, the reason is that operations which it has in common with the English furnace can be more easily dealt with und r A. than B.

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Figs. 408 to 411.—The German Cupelling Furnace at Pribram.

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§ 141. The Furnace.—The furnace selected for illustration is the one in operation at Pribram* (Figs. 408 to 411). It differs somewhat from the generally accepted circular form,† and is an improvement on it. Fig. 408 shows the fireplace *a* at the right, and the flue *b* at the left, of the hearth. Fig. 409 represents a horizontal section of one furnace and the fireplace *a'* of a second furnace, the furnaces being built in pairs. The products of combustion pass downward through four separate flues *b*, which unite in one main flue *c*, leading to the dust-chambers. In Figs. 410 and 411 are seen two vertical sections on the same line *EF*, Fig. 410 representing the furnace before, Fig. 411 after, tamping in the hearth. The furnace is built of common red brick, with the exception of the parts that are exposed to the flame, as indicated by the cross-hatching. In the upper part of the foundation and in the side walls small channels *d* and *e* reaching outward are left open. These serve as drains for the moisture. At the back of the furnace are three openings *f*, through which the tuyere-pipes are introduced. At the front is the litharge channel *g*, which can be closed by a sliding door; a cast-iron breast-plate *h* serves as support for the upper hearth *i*. The movable arched roof *k* rests on an L ring *l*, and is removed with a differential pulley suspended from a traveler. On the foundation is built a brick bottom *m*, the brick being set dry. Beneath its lowest point, just below the cavity *n*, is a cast-iron plate *o*, to prevent any leakage of metal through the drain *d*, should the working-hearth crack or be injured in any other way. The flues leading from the fireplace *a* to the hearth are shown in *j*. The firebrick part of the furnace is encased in cast-iron plates that have openings corresponding to the upper drains. The whole furnace is well bound together by buckstays and tie-rods. The fuel used is a mixture of bituminous coal and lignite; the ash-pit is closed as the blast is introduced beneath the grate bars.

§ 142. Plattner's Cupelling Furnace.—In this connection may be mentioned the Plattner‡ modified German cupelling furnace. It has the form of a reverberatory furnace; the hearth is

* *Oesterreichisches Jahrbuch*, xxxviii., p. 46; Privates notes, 1890.

† *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xxii., p. 89; *Berg- und Hüttenmännische Zeitung*, 1872, p. 415.

‡ Drawings in Arche, "Die Gewinnung der Metalle." Leipsic, 1888, Heft i., plate i.; Schnabel, "Metallhüttenkunde." i., p. 589; Results in *Berg- und Hüttenmännische Zeitung*, 1886, p. 211.

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rectangular in plan (13 by 8½ ft.), and receives its blast from two pipes on either side of the firebridge; the litharge channel is at the opposite end, beneath the flue, which carries off the products of combustion as well as the lead fumes. Less fuel is required and less metal volatilized, because the litharge need not be heated to such a degree to remain liquid; and no fumes enter the cupelling room, as they are all carried off with the fuel gases. Most of the English cupelling furnaces erected at present are constructed on the Plattner model.

§ 143. Mode of Conducting the Process.—The operations necessary to work a charge are six in number: preparing the working bottom, charging and firing the furnace, softening the bullion, cupelling the softened bullion, removing the crude silver, and refining the crude silver.

1. *Preparing the Working Bottom.*—To be suitable for preparing the hearth, the material must be one that will not be attacked by the litharge, nor crack, must be sufficiently porous to absorb some litharge, and free from any reducing agent (organic matter, metallic sulphide). The material most used is a marl. The composition of that used in the Harz Mountains varies, according to Kerl and Wimmer,* within the following limits: SiO_2, 21.22 to 22.24%; Al_2O_3, 5.39 to 6.76%; Fe_2O_3, 3.54 to 5.39%; CaO, 65.65 to 66.41%; MgO, 1.05 to 2.22%. Usually the hearth material is a mixture of dolomite or limestone with fire-clay. For instance, at Tarnowitz† a dolomite of the composition, SiO_2, 6.00%; Al_2O_3, 7.60%; Fe_2O_3, 4.10%; $CaCO_3$, 49.86%; $MgCO_3$, 32.82%; is mixed with 25% of clay. At Pribram three parts of limestone are ground together with one part of clay so as to pass a 5-mesh sieve; at other places, an 8-mesh sieve. The hearth material has to be moistened before it is beaten down in the furnace. For this purpose it is spread on the floor, sprinkled with water from a hose, and turned over and over with a shovel, that the moisture may be equally disseminated through the powder. If left over night, it must be covered with wet cloths and worked again the next day. The material is of the right consistency if, when pressed in the hand, it coheres to a lump, but has not sufficient moisture to adhere to the hand. To obtain a uniform material, it is sifted, just before using, through a coarse hand-

* *Berg- und Hüttenmännische Zeitung*, 1853, p. 241.

† *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, xxxii., p. 107.

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sieve, and any lumps that may have formed are broken up or thrown aside. Sometimes it is introduced all at once, sometimes in two separate layers. The latter is the way at Pribram, as Fig. 411 shows, *i* being the upper and *i'* the lower bottom. Before the filling is put in the brick bottom is sprinkled with water, that it may not take up any moisture from the hearth material. At Pribram the lower bottom *i'* is first tamped down to the form shown in the drawing. The tool required, the tamping-iron, is a cast-iron disk of about 6 in. in diameter and 1 in. in thickness, with a socket into which fits a wooden handle about 4 ft. long. The tamping is begun at the center, proceeds in the form of a spiral to the side walls, and returns in the same manner, care being taken that the circular indentations shall overlap in part those made in working outward. By giving attention to this point, the surface will be evenly beaten down, which is essential. Before putting down the upper bottom, the surface of the lower one is roughened by scratching it with the point of a chisel. This is done that the bottom to be put down may adhere to the one already in place. The tamping of the upper bottom is done in the same way as the lower one, only the surface, when finished, must be perfectly smooth. It is of prime importance that the hearth should have just the right degree of hardness. This is easily indicated to the ear and hand after a little experience. If too hard, it will crack and not be sufficiently porous; if not hard enough, it will absorb too much lead. If the material was too dry, the hearth will peel when heated; if too wet, it cannot be beaten to the desired hardness. In order to prevent its adhering to the tamping-irons, these are slightly warmed. The thickness of the hearth at the bottom and sides varies somewhat; the least is perhaps 6 in. at the bottom and 8 in. at the sides. The general rule for the curvature of the hearth is that the more concave the bed is the easier will be the cupelling and the harder the finishing; the flatter the bed, the harder the cupellation and the easier the finishing. When the hearth is completed, a cavity *n* (1 in. deep and 34 in. in diameter) is cut in the deepest part to receive the silver. It is located a little to one side of the medial line, toward the firebridge, that the silver may be kept easily molten at the end of the operation.

2. *Charging and Firing the Furnace.* — The furnace now receives its charge of 25 tons of lead, enriched by the Luce and

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Rozan process. Sometimes the bottom is covered with straw before charging, to prevent its being damaged during the operation. The bars are placed in such a way as to leave an open space reaching from the tuyeres to the litharge channel. The hood is then lowered on a clay lute placed on top of the furnace. The litharge channel is closed by lowering the door, the fire kindled on the grate, and soon the blast below let on.

3. *Softening the Bullion.*—The lead melts down slowly. The dross rises to the surface and is drawn off through the litharge channel. The temperature is raised and the blast put on through three tuyeres, the skimmings form and are drawn off, and finally pure litharge takes their place. At the very beginning, when the dross is being removed, bubbles of gas are seen to pass off from the surface of the lead and this continues through almost the whole cupellation. They are mainly water-vapor and are mixed with carbon dioxide, resulting from the decomposition of the magnesium and calcium carbonate of the hearth material.

4. *Cupelling the Softened Bullion.*—The temperature and the blast are now lowered, and are kept low during the larger part of the operation. They are raised only toward the end, when the enriched silver-lead alloy requires a higher temperature to give up the last parts of lead. When the skimmings have been removed and the cupellation has somewhat progressed, the convex surface of the lead will be exposed to the action of the blast, while the lead near the periphery will be covered with litharge. The width of this rim depends on the rate at which the litharge is allowed to run off through the litharge channel. As litharge melts only at 954° C. (§ 5), a temperature of about 980° C. has to be maintained, if it is to remain liquid. Lead, melting at 325° C. (§ 4), would be volatilized to a considerable extent, if fully exposed to the action of the blast at this high temperature. The litharge is therefore allowed to run off only to such an extent as to give the rim a width varying from 12 to 15 in. at the begining, and of 5 in. toward the end of the operation. The blast playing on the surface of the lead forms small waves and drives the litharge toward the channel. The pressure is about 8 oz. per sq. in., and about 300 cu. ft. of air are delivered per minute through the three tuyeres. In order to remove the litharge, a gutter is cut into the channel by means of a scraper. This is an iron rod, 8 ft. long and $\frac{3}{8}$ in. in diameter,

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flattened out at both ends. One of these is bent to encircle a wooden handle, while the other, only slightly flattened, is sharpened and bent to the form of a hook. In cutting the gutter, the entire edge of the tool must be used, and not one of the corners. If larger pieces of the breast are to be cut out, it is done with a chisel-pointed bar, say ¾ in. in diameter. The rate at which the litharge runs off depends on the depth of the gutter and the strength of the blast. The depth is correct when the litharge runs off in a thin stream which stops as soon as the blast is lowered. If it runs too fast, the rim of litharge in the furnace decreases and the lead is volatilized, while the litharge is not sufficiently desilverized from too short a contact with the lead beneath it. If it runs too slowly, the rim of litharge becomes too broad and the cupellation is retarded; there is again loss in silver by the higher temperature that is required to keep the larger amount of litharge liquid, and if the temperature be not raised sufficiently, lead will be carried out mechanically by the litharge. The litharge gutter is first cut into the breast on the side farthest away from the bridge and gradually moved toward the opposite side, that the final litharge may be drawn off as near the firebridge as possible. The current of the litharge, when flowing out of the furnace, is directed in such a way as to form a large cake in front of the furnace. In some instances a U-shaped piece of sheet-iron is placed upright in front of the channel, that the litharge may collect in a rectangular block. Before removing this, the litharge in the center that is still liquid is tapped from near the bottom of the cake. The litharge resulting from the cupellation is graded according to the silver contents and the percentage of impurities (copper, bismuth, etc.), into marketable litharge and into a by-product to be treated by a separate process. As the cupellation progresses toward the end, the temperature is raised and the blast increased, the side tuyere-openings are closed, and two tuyere-pipes introduced through the central opening; finally, the last film of litharge disappears from the surface of the lead with a characteristic phenomenon, the so-called brightening,* which every reader has watched while making a silver assay.

The products of the Pribram cupelling furnace have, according to Dietrich, the following percentages of composition:

* Van Riemsdijk, *Berg- und Hüttenmännische Zeitung*, 1880, pp. 247, 275, and Bock, *Op. cit.*, 1880, p. 409, have made interesting investigations on this subject.

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	Dross.	Tin Skimmings.	Antimony Skimmings.	Red Litharge.	Green Litharge.	Cupel Bottom	Flue Dust.
Pb	30.75	13.40	.....	.....	.....	.....	.....
PbO	55.27	64.97	77.95	98.370	98.140	68.860	64.41
PbO combined with $(AsSb)_2O_5$	.....	.....	11.87	.....	.....	.....	.....
CuO	1.99	0.29	0.28	0.069	0.080	0.070	.....
Bi_2O_3	.....	.....	.....	.....	.....	Trace.	Trace.
MoO_4	.....	.....	.....	.....	.....	Trace.	.....
As_2O_5	1.42	1.87	0.92	0.010	0.009	.....	Trace.
Sb_2O_5	1.83	6.76	5.85	0.074	0.067	0.530	11.40
SnO_2	0.72	10.31	Trace.	.....	.....	.....	.....
Ag	.....	.....	0.004	.....	.....	.....	.....
Ag_2O	0.307	0.189	.....	0.004	0.0048	0.170	0.013
Al_2O_3	0.54	0.23	0.32	0.072	0.056	2.120	.....
Fe_2O_3			0.14	0.010	0.014	0.300	.....
ZnO	0.13	0.05	Trace.	0.009	0.012	Trace.	0.50
Ni	0.09	Trace.	.....	.....	.....	.....	.....
NiO	.....	.....	0.04	0.005	0.005	.....	.....
CaO	0.45	0.43	0.95	0.256	0.362	.....	Trace.
$CaCO_3$	.....	.....	.....	.....	.....	24.100	.....
MgO	.....	.....	.....	.....	.....	Trace.	.....
CO_2	.....	.....	.....	0.383	0.432	.....	.....
SiO_2	1.75	0.94	0.87	0.320	0.350	2.970	4.35 + ash.
S	2.80	0.65	0.09	.....	.....	.....	.....
SO_3	4.12	1.37	.....	0.084	0.027	0.040	16.65
C	.....	.....	Trace.	.....	.....	.....	Trace.

CRUDE SILVER.

	Pribram. (*a*)	Freiberg. (*b*)	Wyandotte, Mich. (*c*)	
Ag	95	92.190	98.6910	99.5980
Pb	5	4.210	1.0900	0.2600
Cu	.....	2.104	0.1170	0.1060
Ni, Co	.....	0.600	0.0040	0.0060
Fe	.....	.....	0.0900	0.0310
Bi	Trace.	.....	0.0058	.....
Au	Trace.	.....	0.0023	0.0015

(*a*) *Loc. cit.* (*b*) Stölzel, "Metallurgie," Brunswick, 1863–66, p. 1182. (*c*) "Transactions of American Institute of Mining Engineers," ii., p. 97.

The time required to cupel the 25 tons of rich lead is 80 hours. It is divided as follows:

	Hours.
Preparing the hearth-material, making the hearth, and charging the lead	8
Melting down and wheeling the necessary coal	16
Drossing	6
Drawing the tin skimmings	6
Drawing the antimony skimmings	3
Running off market litharge	23
Running off rich litharge	18
	80

The cupellation is in charge of three men, each with a helper, working in eight-hour shifts. For every 100 tons of base bul-

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lion are consumed 19.63 tons of coal and 23 bushels of hearth material (which includes the refining of the silver). The loss in silver is 0.83%; that in lead, 4.33%. The figures do not include the loss endured in re-treating some of the by-products. Thirty-six per cent. of the litharge is low enough in silver to be sold in the market, the bullion from which it is made averaging 167 oz. silver to the ton.

5. *Removing the Crude Silver.*—After the silver has brightened, the blast is shut off, the tuyere-pipes are removed, and the litharge gutter is closed with a ball of clay. Two knife-shaped pieces of wrought iron are introduced through the litharge channel and pressed into the cake of crude silver. First warm, then cold, water is allowed to run into the furnace, and the silver then removed through the central opening at the back. It is cleaned, weighed, etc. The furnace is left to cool till the next day, when the hearth is examined for small particles of silver that have adhered to it. The upper hearth is then removed with a pick. Part of it is soaked with litharge to a depth of 2 or 3 in. This is separated from the unsoaked part, which is mixed in with the hearth-material for the next charge, while the lead-soaked part goes to the blast furnace.

6. *Refining the Crude Silver.*—The refining of the crude silver has for its object the removing of impurities, which vary from 2 to 10%. Formerly this was done exclusively in a small oval reverberatory furnace, having a working door at one side or at the flue-end, and a working bottom of similar composition to the cupelling bottom, the reason being that the loss in silver and the consumption of fuel were considered very much smaller than if the firing was done in the big cupelling furnace. Ohl* and Foehr† have since proved this not to correspond to the facts, and the refining in a separate furnace has received a check. Since the discovery by Roessler‡ how to refine crude silver in a plumbago crucible by means of silver sulphate, the refining in a separate furnace has been abolished entirely at some works.

Refining in a crucible is carried on at small works by melting down the silver, uncovering the crucible that the air may oxidize the impurities. These are stiffened by sprinkling bone-ash or

* *Berg- und Hüttenmännische Zeitung*, 1879, p. 274.
† *Ibid.*, 1885, p. 381.
‡ *Ibid.*, 1889, p. 387.

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hearth material on the silver and then removing it with a skimmer, the operation being repeated till no more impurities rise to the surface. A slag obtained by Curtis* at Wyandotte, Mich., sand being used in refining, contained, in addition to silicate of lead, the following metals: (NiCo) O, 0.550%; CuO, 0.203%; Bi_2O_3, 0.026%; Ag, 1.837%; Sb_2O_4, 0.639%; As_2O_3, 0.005%—total, 3.260%.

Roessler found that if silver sulphate is added to melted silver in a crucible, first the lead and then the bismuth are converted to sulphates, the silver being at the same time set free. Copper is not removed by silver sulphate. By keeping separate the different slags he concentrates the bismuth in a comparatively small amount of slag, to be treated separately, while the first slag contains most of the lead. To prevent the crucible from being attacked, he introduces a layer of quartz-sand on top of the silver, and then stirs in the silver sulphate in the middle. The sand serves at the same time to stiffen the slag, which is then removed with a skimmer. The process, as seen by the writer at the Lautenthal Smelting and Refining Works in 1890, differs slightly from the manner indicated above. It is as follows: The silver sulphate is produced by dissolving silver in sulphuric acid of 66° B. in a small cast-iron kettle. The solution is allowed to cool, is then diluted to 60° B., when nearly all the silver sulphate will fall out as a slightly yellow cheesy mass. The supernatant liquor is drawn off as much as possible, and the remaining dilute acid driven off by heating. Special arrangements are required to cool the vapors, as they carry finely divided silver sulphate along with them. The temperature is raised to redness in order to fuse the silver sulphate, which when liquid is cast into molds and is ready for use. The color of the melted sulphate is grayish-green; it is hygroscopic, and is therefore kept in a lead-lined wooden box; 1,000 parts contain 650 parts of silver.

Crude silver of a fineness varying from 950 to 980 thousandths is melted down in a plumbago crucible holding 700 lb. The crucible is heated with coke in a small cylindrical furnace having in the lower part two 1¼-in. openings for the blast-pipes. On trying to stir in the silver sulphate, as advised by Roessler, it was found that sometimes it got beneath the layer of sand, spread over the

* "Transactions of American Institute of Mining Engineers," ii., p. 98.

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silver, and corroded the crucible. To prevent this, a wrought-iron ring ($\frac{1}{8}$ in. thick, $10\frac{1}{2}$ in. in diameter, and 7 in. high) is coated on either side with a 3-in. layer of clay, and placed on the silver. Into the center are introduced with a ladle from 6 to 8 lb. of sulphate (the size of a hen's egg) that has been warmed. As soon as it comes in contact with the silver this begins to boil. When the effect decreases, the silver is stirred with an iron rod to assist the action of the sulphate. From 25 to 30 minutes after the sulphate has been added this is completely decomposed, a slag has collected on the surface of the silver, and quartz is added to stiffen it, that it may be removed with a skimmer. A second, a third, and, if necessary, a fourth addition of silver sulphate is given to make the silver fine. The test made for fineness is to dissolve some silver in nitric acid and to supersaturate with ammonia. No precipitate must form even after standing.

The amount of silver sulphate required to fine the silver is about $1\frac{1}{2}$ times the total quantity of base metal present. Thus 700 lb. of crude silver, being 970 fine, contain 21 lb. of base metal, which would require 31 lb. of silver sulphate to be added in three portions. If the test with ammonia should prove this not to be sufficient, an extra addition is made. In 1890, 107,031 lb. of crude silver with an average fineness of 970 required 6,009 lb. of silver sulphate, which corresponds to about 2 parts of sulphate to 1 part of base metal. All the silver of the sulphate is not taken up by the silver in the crucible; part of it enters the slag, as shown by the following analysis made by Hampe:*

SiO_2, 40.7%; P_2O_5, 0.64%; SO_2, 0.61%; S, 0.15%; FeO, 13.47%; Al_2O_3, 0.43%; Bi_2O_3, 6.01%; PbO, 33.50%; Ag_2O, 2.05% (=1.88 Ag.); Cu, 0.45%; Sb, 0.02%; CaO, 1.73%; MgO, 0.25%; K_2O, 0.64%; Na_2O, 0.26%.

The main advantage of Roessler's method of refining is to be found in the larger direct output of silver and the concentration of the bismuth in a comparatively small amount of slag that is more easily worked than cupel-bottom and litharge obtained in the reverberatory used for refining the silver.

B. English Cupellation.

§ 144. Characteristics.—The characteristics of this method are a small reverberatory furnace with a movable bed and a fixed

* *Berg- und Hüttenmännische Zeitung*, 1891, p. 187; *Engineering and Mining Journal*, Nov. 14, 1891.

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roof, and the fact that the bullion to be cupelled is charged gradually and the silver refined in the same furnace where the cupellation was carried on.

§ 145. The Furnace.—This has undergone many changes from the original English furnace as described by Percy.* Figs. 412 to 416 represent one form of cupelling furnace that is commonly

Fig. 412.

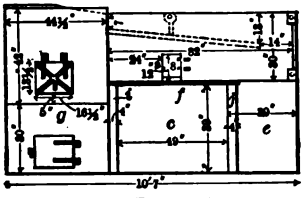


Fig. 413.

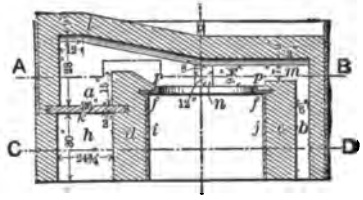
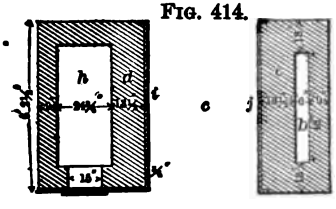


Fig. 414.



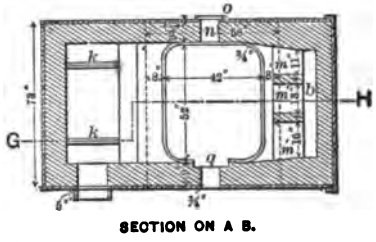
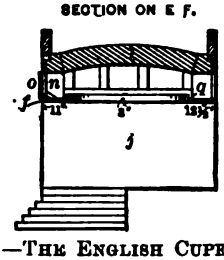


Fig. 415.

Fig. 416.



Figs. 412 to 416.—The English Cupelling Furnace.

used in American refining works. The vertical section (Fig. 413) shows the general construction of the furnace, with the fireplace *a*, the vertical flue *b*, to the right, and the space *c* between the firebridge wall *d* and the flue wall *e*. This is closed at the top by the compass-ring *f* and the test when it has been put in place. The upper part of the furnace is encased in cast-iron plates; the side castings have a strengthening rib to resist the

* "Metallurgy of Lead," p. 178.

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thrust of the roof. In addition, the front *g* of the ash-pit *h*, and the flue-end of the furnace, as well as the inner sides of *i* and *j*, are protected by castings. The usual buckstays and tie-rods have been left out in the drawings. To be noted are the large grate area (4 ft. 6 in. by 2 ft. ¾ in.) in comparison with the hearth area (4 ft. 4 in. by 3 ft. 6 in.), the height from the grate bearer *k* to the roof *l* (2 ft. 4 in.) and the short distance (9½ in.) between the roof and the top of the compass ring, as they are all essential for good working. The flame from the grate-bars is directed downward by the pitch of the roof, and, being forced to pass through the small space between the shallow roof and the hearth, exerts all its heating force on the lead. By paying special attention to this part of the construction it is possible to obtain a sufficiently high temperature to refine silver without being forced to use special kinds of bituminous coal. In fact, with undergrate blast, slack coal is good enough for cupelling, and nut coal is required only for fining, the coal being of ordinary grade.

The grate in the drawing is so arranged that firing, as well as stoking, is done from the front. Another plan is to place the grate-bars parallel with the ends of the fireplace, and to stoke from the side of the furnace. If in such a case forced blast is used, the casting at the stoking side has two oblong openings, each to be closed by a cast-iron door, say 21 in. long and 6 in. high, having its two hinges on the lower side. The horizontal flue *m* (Fig. 413) is seen in Figs. 415 and 416 to be divided into three smaller flues *m'*, increasing in width (11, 13, 16 in.) from the back of the furnace toward the front, the object being to prevent the flame from taking the shortest line toward the center of the flue and to draw it somewhat toward the front, whereby the litharge floating on the front part of the lead is kept hot. At some works this flue is divided into five or six smaller flues. If the conditions of the draft are such that the flame rushes too much toward the center, this is remedied by placing one or more firebricks in the flue, which will correct the evil. At the back of the furnace there is only one door *n*, through which passes the blast-pipe and through which is fed one pig of lead at a time. Many furnaces have three small openings, a central one for the blast-pipe and two lateral ones, through each of which a pig of lead is gradually pushed forward and melted away. Having

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only one opening, to be closed by a sliding door running in a vertical frame *o*, simplifies the casting. The compass-ring, whose inner contour must, of course, correspond to the form of the test that is to pass through it, is intended for a rectangular cast-iron test having rounded corners. The upper rim *p* serves to hold in place the brickwork with which its surface is covered. It extends all around the central opening, with the exception of the front, where it stops for a distance of 16 in., leaving room for the slot *g* (4 in. wide), through which the litharge is to run down into the litharge-pot. The outer contour of the compass-ring has the rectangular form of the furnace, and reaches from front to back. It thus forms at the front the support for the working tools, and at the back the support for the prop which gives the blast-pipe the desired pitch. At the front the compass-ring is often left open the entire length of the litharge-slot. In such a case the support for the tools is a special cross-bar (skimming-bar) held in place by screws, or by being let into the brickwork, or fastened in some other convenient manner.

Another form of furnace, constructed according to the Plattner principle, is shown in Figs. 417 to 425. The side elevation (Fig. 417) and plan (Fig. 418) show the general construction of the furnace. From the grate *a*, worked with undergrate blast, the flame passes through a narrow flue *b* (3 ft. wide) on to and across the hearth *c* (5 ft. wide, increasing to 5 ft. 3½ in.), ascends in the vertical flue *d* (2 ft. 6¾ in. wide), returns through the horizontal flue *e* (23 in. square), and then descends through *f* (of same size) into a main flue underground. Air for cupelling is delivered on to the hearth by the water-cooled tuyeres *i* and *i'* on either side of the firebridge. The brick-work of the fireplace is protected on the hearth side by a heavy casting *g*, the other three sides are strengthened by horizontal rails *h* and *h'* laid in the brick-work and secured by buckstays and tie-rods. The side walls of the hearth are inclosed by a heavy casting *j*, the flange of which *q* (Figs. 417, 418 and 420) supports them and forms at the same time the compass-ring for the test. The back of the casting is let into the brick-work of the fireplace, its sides are supported by the buckstays *k*, and its front by the legs *l*. The flue *e* rests on iron plates *m*, which are supported by longitudinal rails *n*; these are carried by the transverse rails *o*, which are bolted to the flanges of two pairs of buckstays *k*, and by the

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brick-work of the flue *f*. The flue is bound with angle-irons and tie-rods. The test (Figs. 417, 418, 419, 421 and 422) consists of an oval ring *r* of boiler-iron, strengthened at top and bottom by flanges *s*, and bound at the front by a loop-shaped band of iron. It rests on a cast-iron bed-plate *t*. Its manner of support is shown in Figs. 424 and 425. Two rails *AA'* run along the bottom of the test; at one end they rest on the rounded bar *B*, supported by a pair of jackscrews, at the other they are held in place by the looped rods *RR'* suspended from the yoke *Y*, which is connected with a second yoke *Y'* by means of the right and left hand-screw *SS'*. By turning the wheel *W* the front of the test can be raised and lowered without difficulty. Into the loop at the front fits the cast-iron breast-jacket (Fig. 423), which rests on the cast-iron bed-plate *t* (Figs. 417 and 419). The test is not water-cooled, but in order to protect the filling material from the corrosive action of the litharge, a pair of water-pipes (*w* and *w'*, Fig. 419) is imbedded in it.

A furnace similar to the one just given and used for the concentration of matte has been previously described and illustrated (page 371). The test is water-cooled and can easily be raised and lowered while working. It is very well suited for cupelling; if used for this purpose the test ought to be made shallower.

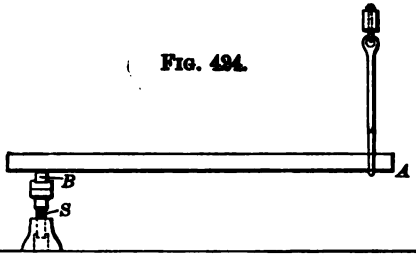
In one instance the English cupelling furnace has been so modified as to have the fireplace at the end of the elliptical test, and two converging blast-pipes at the side.

§ 146. Test Rings.—The test (4 ft. by 2 ft. 6 in.), which originally consisted of an oval frame of wrought-iron (4½ in. wide, ½ in. thick) filled with bone-ash, has undergone many changes in construction, manner of support, and filling material. A few of the tests in use at present are represented in Figs. 426 to 435. Figs. 426 and 427 show a cast-iron test *i* resting on a test carriage. It has a concave bottom and a cast-iron pattern of the cavity on the inside, both of which will be discussed further on. The test ring is rectangular in plan and has rounded corners. It fits into the cupelling furnace shown in Figs. 412 to 416. When in place its upper surface will be on a line with the upper rim of the compass ring *p* (Fig. 413). In front it has a 3-in. slot *j* which is closed when the filling material forming the hearth is being rammed in. The rectangular form of test offers a large

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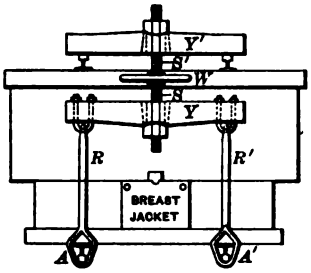
Support of Concentrating Test.

Fig. 424.



SIDE ELEVATION

Fig. 425.



FRONT ELEVATION

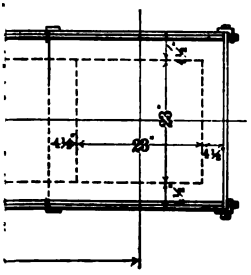
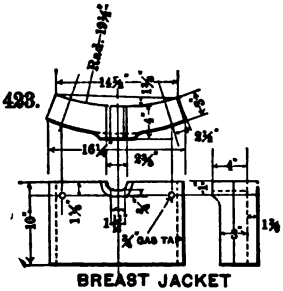
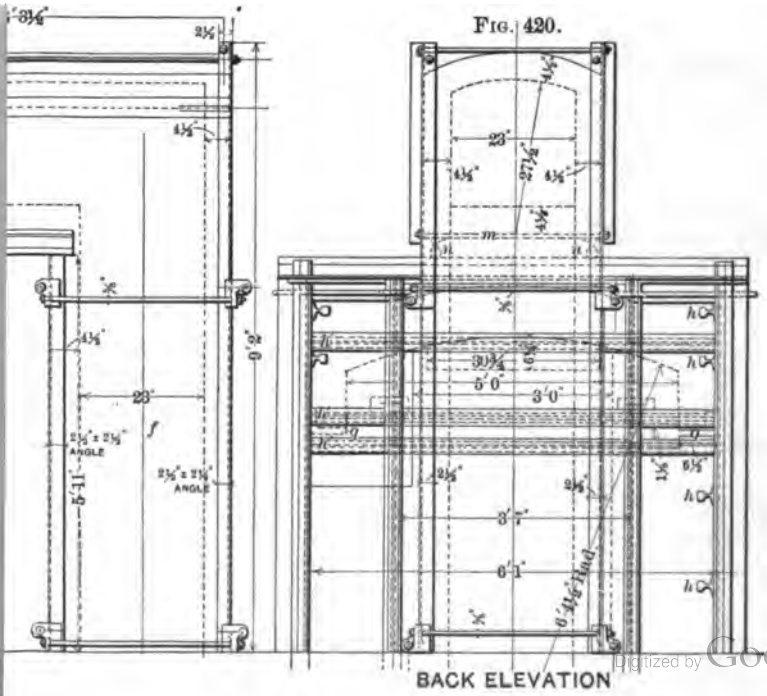


Fig. 423.



BREAST JACKET

Fig. 420.



BACK ELEVATION

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FIG. 426

SECTION ON A B.

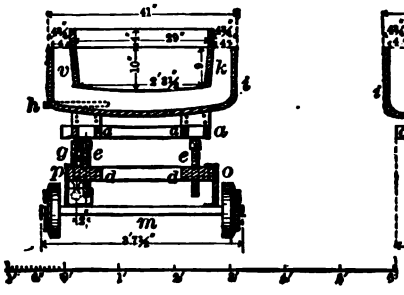


FIG. 427.

SECTION ON C D.

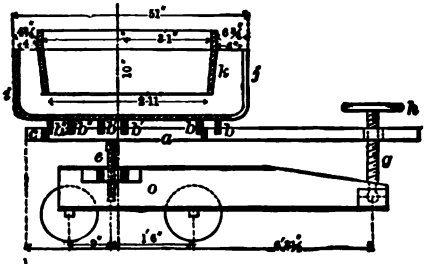


FIG. 428.

BOTTOM DOWNWARDS

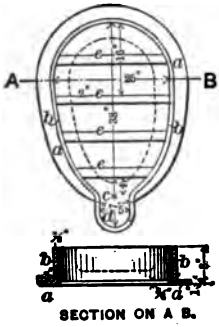


FIG. 429.

BOTTOM UPWARDS

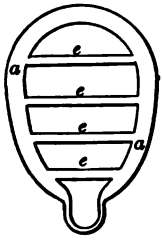


FIG. 430.

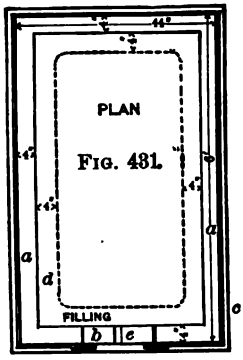


FIG. 431.

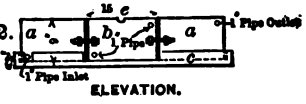


FIG. 432.

FIG. 433.



FIG. 434.

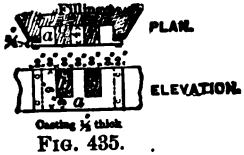


FIG. 435.

FIGS. 426 TO 435.—TEST RINGS.

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surface for oxidation, and therefore more lead can be cupelled on it than on an oval hearth.

Figs. 428 to 430 represent an oval cast-iron test ring, having a horizontal flange *a*. When in place, the uper side of this will be close to or in contact with the lower side of the compass ring *f* (Fig. 413), and the upper edge of the test ring itself will be in line with the upper edge of the compass ring, thus making the distance between the roof and the surface of the lead as small as possible. The test ring protrudes over the horizontal flange at *c*, forming a loop *d*. In tamping down the filling material the loop is not filled, in order that the litharge overflowing from the hearth may pass through it into the litharge-pot below; thus any contact between hot litharge and iron frame is avoided. Across the bottom of the test ring are four cast-iron arms *e* to hold the filling in place.

With both tests the corrosive action of the litharge very soon eats out the filling, especially at the front, and in a comparatively short time the test has to be removed from the furnace and replaced by another. To counteract the corrosion of the front, and at the same time permit the raising or lowering of the litharge gutter, a water jacket, as shown in Figs. 434 and 435, is fastened by means of bolts to the test ring, be this cast iron or wrought iron. The jacket has the same depth as the test ring and forms the breast. The litharge runs off through gutters cut into the filling *b* (Fig. 435). As this is cooled by the water circulating in the jacket, it is eaten out only very slowly. The jacket itself does not come in contact with the hot litharge in the furnace, as it is protected from it by a 3-in. rim of filling. This wears out somewhat, but never or rarely so far as to bring the casting into direct contact with the hot litharge.

By this arrangement only the front is protected. A device that protects the sides alone is shown in Fig. 433. Here *a* is a cast-iron test ring, resting on a bed-plate *b*. It is surrounded by an iron hoop tied by an iron band. On top of the test ring are placed two 1-in. pipes in which water circulates. The filling *d* is rammed down in the usual way. The lead, while the furnace is running, being always kept about at the same height, the litharge can show its bad effect only on the level of the water-pipes, and these effectively counteract to a great extent the corrosive action.

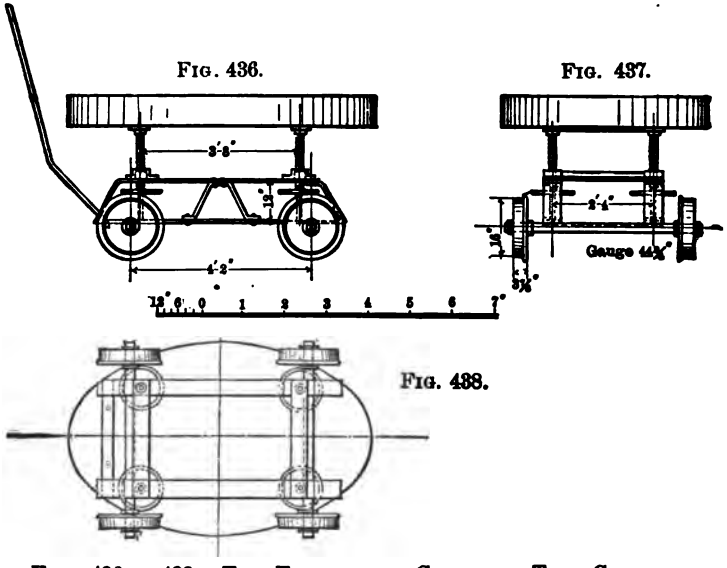
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A combination of the two arrangements is shown in Fig. 419. Going one step further in water-cooling we come to the Steitz water-jacket test, as represented in Figs. 431 and 432. Here *a* is a rectangular water jacket made of boiler iron. The open space in front is closed by a cast-iron or bronze water jacket having a litharge gutter *e*; it is fastened with bolts to the wrought-iron jacket. The jackets are placed on a cast-iron test plate *c*, which supports the filling *d*. Here both jackets are protected from the hot litharge forming on the surface of the lead in the furnace. The gutter *e* alone is attacked by the litharge, and is eaten out after some time; the breast jacket then has to be exchanged for another. This can be done quickly, and while the furnace is being used. The effect of jacketing is that the filling lasts longer than with the other test rings. The cast-iron front jacket, in addition to preserving the breast better than any of the other arrangements, has for concentrating another advantage, that the depth of the litharge gutter does not have to be regulated by the cupeller, but always remains the same. This renders it, however, unavailable for bringing very rich, say 70%, bullion up to fine silver, as the uniform level of the gutter prolongs indefinitely the removal of the last lead contained in the silver.

§ 147. Test-Ring Supports.—The manner of bringing the tests into position and fixing them there has of late years undergone various changes. The old method consisted in driving four iron wedges between the bottom of the test ring and two transverse bars, the ends of which were inserted 4 in. below the test frame into the firebridge wall *d* and the flue wall *e* (Fig. 413). It is not much used now.

Sometimes wedges are still retained to adjust a large test frame, as represented in Figs. 431 and 432. This is placed on two 9-in. brick walls running along the firebridge wall and the flue wall. It is then raised gradually to the height of 12 in., and four pillars, each three bricks high, are placed beneath the bed-plate, which brings it up nearly to the compass ring. By then driving flat wedges between the bed-plate and the brick posts the test is adjusted to its final position. The usual way, however, is to use jackscrews, which are satisfactory if the entire hearth is made accessible as in the modern cupelling furnace where the hearth walls rest on cast-iron supports. A common

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Figs. 436 to 438.—The Fraser and Chalmers Test Carriage.

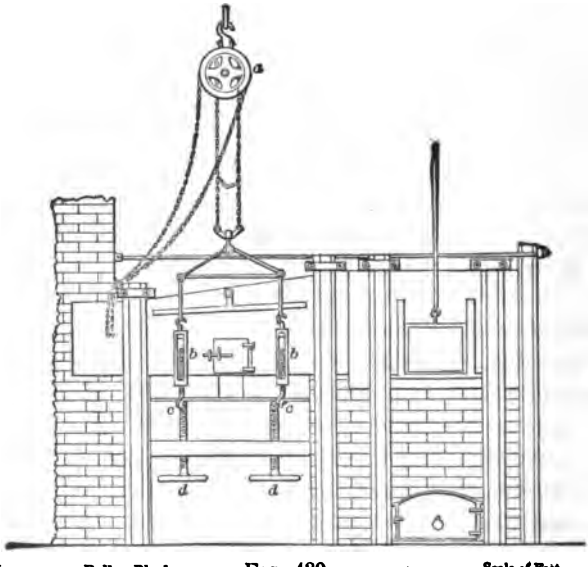


Fig. 439.

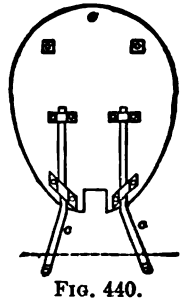


Fig. 440.

a Pulley Block
b Turn Bolts
c Support Bars
d Back Screws
e Test Plate

Scale of Feet

Figs. 439 and 440.—The Lynch Test Support.

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method of supporting a small test is by four screws (18 in. long, 1½ in. in diameter) working in two transverse bars placed 12 in. beneath the test ring. This rests on a cast-iron plate into which the points of the four screws are set.

Four screws are also found in connection with a test carriage, as shown in Figs. 436 and 438.* Here the test is easily brought into position, and then raised by means of the screws and brought up close against the compass ring.

With the three arrangements described, the test, when once placed, is immovably fixed while the cupellation is proceeding. Further, when the furnace has become hot, the turning of the four screws sometimes presents difficulties. These have, however, been largely overcome by more perfect air-cooling. To

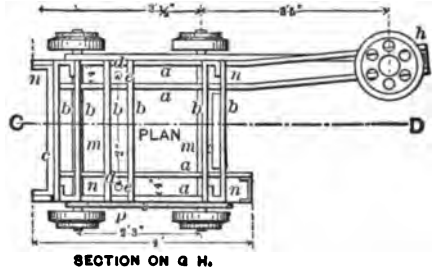


FIG. 441.—THE RHODES TEST CARRIAGE WITH SWINGING TEST.

obviate these disadvantages the two front screws have been removed and the test suspended or held by a support.

The Lynch† test support is represented in Figs. 439 and 440. The test plate *e*, shown bottom side up in Fig. 440, has at the back the two sockets for the points of the two back screws *d* (Fig. 439). Two support bars *c*, diverging 27 in. when extending in front of the furnace, are clamped to the front half of the plate. When in place they are hooked with the turn-bolts *b* to a triangle made of ¾-in. iron, which is attached to a differential pulley *a* hung from the roof. With the pulley the test can be raised or lowered quickly and evenly to regulate the flow of litharge without altering the depth of the litharge gutter. The contents of the test (rich lead or silver) can also be poured. With

* Taken from a drawing of Messrs. Fraser & Chalmers.

† Blake, "Transactions of American Institute of Mining Engineers," x., p. 220; Patent No 275,232, April 3, 1883.

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the turn-bolts it can be moved sideways to counteract the action of the litharge, should this corrode one side of the hearth more than the other.

Another movable test support is the one by F. B. F. Rhodes, represented in Figs. 426, 427, 441. The test is supported by a carriage *o*, with its movable upper frame *aa'*. This rests at the back on two screws *e* and *e'*, working in the blocks *d* and *d'*. At the front, it is supported by the screw *g*, working in the right arm *a* of the frame, which extends a short distance in front of the furnace, and is turned outward sufficiently for the wheel *h* to be to the right of the cupeller. By inserting a hook into one of its circular openings he turns the wheel to right or left, and thus raises or lowers the front of the frame *aa'* and with it that of the test. Into the upper frame cross-beams *b b' b''*, riveted to each other in pairs, are let in, which serve as support for the concave bottom of the test. If this were straight the surface of beam *a* would represent in the section an unbroken line, two more beams like *c* and *c'* being sufficient to give the frame the required stability. The wheels of the carriage are grooved. By having two rails running across the space *c* (Fig. 412) all shifting of the carriage to right or left to get it into correct position is avoided.

The support at the back by the screws *e* and *e'* has, at least with the old cupelling furnace, the disadvantage that when the frame *aa'* has to be lowered there before the carriage is taken out, much difficulty is experienced in turning the screws. To avoid this the screws *e* and *e'* have been replaced by two pivots which, being fastened into the blocks *d* and *d'*, fit into circular openings of two blocks fastened to the beams *a* and *a'*. When the frame, resting at the back on these two pivots and supporting the test, has been wheeled into its correct position and is to be raised, this is done by lifting it a few inches with a crowbar; then two U-shaped castings, of the correct height, and wide enough to inclose the pivots, are placed around these. On withdrawing the crowbar the frame sinks on these two castings, which then support it. When the test is to be exchanged, the frame is again slightly raised, the two U-shaped castings are thrown off, and the frame is slowly lowered, when it will again be supported by the pivots. The test ring is then sufficently low for the carriage to be withdrawn from under the compass ring.

A movable test support, manufactured by the E. P. Allis Co., Milwaukee, Wis., was described on page 371.

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Another movable support with specially constructed test ring has been invented by Roesing,* and is used at Tarnowitz, Silesia. Movable tests are in much favor when the test ring is not water-cooled, *i.e.*, where the level of the lead is gradually lowered by cutting deeper the litharge gutter; with test rings that are entirely water jacketed they have not been used until lately.

Whatever test support may be in use, care should always be taken to plaster over with clay the upper surface of the ring, so as to protect it against coming in direct contact with the flame, and at the same time to prevent the air from rushing in between compass ring and test ring.

§ 148. The Blast.—The blast was originally produced by a steam jet. This has given way entirely to a blower. The machines in common use are the Baker and Root blowers, and the Sturtevant fan; blower and engine are usually supported on the same bed-plate. The pressure of the blast is about 4 oz. per sq. in. The blast-pipe is of sheet-iron and about 3 in. in diameter. It is frequently fitted into a cast-iron nozzle, sometimes water-cooled, which delivers the air through an aperture 4 in. long and ½ in. wide. This is better than simply flattening the end of the sheet-iron pipe.

§ 149. The Tools.—The tools required by the cupeller are few: two rods (7 ft. long, of ½-in. iron), bent to a hook; one chisel-pointed bar (7 ft. long, of ¾-in. iron); one saw (9 in. long, ½ in. wide, 2 in. deep), attached to a handle (6 ft. long, of ½-in. iron), to cut down the breast; one fire-hook (10 ft. long, of 1-in. iron, with a 4 by 10-in. head); one scoop; one ladle; 10 bullion molds or silver molds, and three or more litharge buggies having small kettles (13 in. in diameter and 8 in. deep) to hold the litharge.

§ 150. Mode of Conducting the Process.—The operations include filling the test and putting it in place, cupelling, and refining.

1. *Filling the Test and Putting it in Place*—The material originally used to fill the test was bone-ash ground fine enough to pass a 26-mesh sieve. This, being too expensive, has given way to a mixture of limestone and fire-clay ground through a 12-mesh sieve, the proportions being three parts by volume of

* *Berg- und Hüttenmännische Zeitung*, 1883, p. 577; *Engineering and Mining Journal*, Jan. 19, 1884.

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limestone to two, or sometimes only one, part of clay, according to the plasticity. Portland cement is used at some works, and if of best quality is more durable than the limestone-clay filling. Sometimes a mixture of two-thirds cement and one-third coarsely ground firebrick is used instead of pure cement. The test has also been brick-lined; magnesia-brick, which resist the corrosive action of litharge better than any other material, are not much used on account of the great cost.

In filling the test-ring with the limestone-clay mixture, this is moistened, as shown in § 143, and then tamped into the ring. Some works beat down the mass in three separate layers; others add the necessary material all at once and begin then with the tamping. If the cast-iron test ring (Figs. 428 to 430) is to be filled, a piece of wood having the form of the loop *d* is put in place, and then the filling beaten down. When finished, the wood is taken out, leaving open the slot for the discharge of the litharge. The wood is best withdrawn only when the test has somewhat dried, as then there is less danger of breaking off part of the filling. The tamping-irons are about 4 in. in diameter at the base. The test is filled entirely with the material, and the cavity then scooped out with a trowel. A very good way to insure a uniformly hard surface is to place a wooden frame on top of the test ring, and then partly fill it with the hearth material. When the frame has been removed, the excess of material is scraped off down to the test ring, and the cavity then scooped out. A rim from 3 to 4 in. wide at back and sides, sometimes 6 and even 10 in. at the front, is left untouched. The cavity has its lowest point near the front to facilitate the dipping out of concentrated bullion or fine silver. The filling material should be at least 4 in. thick at the lowest point, and the depth of the cavity about 5 in. Thus an oval test 4 ft. 6 in. long, 2 ft. 1 in. wide, and 5 in. deep, holds about 2,500 lb. of lead.

In filling the test with Portland cement, or with a mixture of cement and ground brick, this is moistened and tamped down in the usual way, the cavity, however, being formed during the tamping. A quicker and better method is to place the test bottom upward over a mold having the form of the cavity, and then to beat down the moistened cement. In using cement it is essential that the work be done quickly, as it must be finished before the cement shows any signs of setting.

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The test frame *i* (Figs. 426 and 427) is the only one that has a solid cast-iron bottom; it is slightly concave. The working bottom is made of firebrick, which is set dry and then thoroughly grouted with fire-clay. To build up the sides, the cast-iron frame *k* is oiled and put in place, and moistened cement is beaten down in the intervening space *v*, to the top of the test-frame; the iron frame is then carefully removed.

When a test is filled, it has to stand for a fortnight and longer in a warm place (usually the cupelling-room) to dry. Before it is to be used in a cold furnace, a small charcoal fire is made on it. In a warm furnace the fire is kept low for three or four hours after the test is in place.

2. *Cupelling and Refining.*—When the test is in place and well warmed, the temperature of the furnace is gradually brought to a dark-red, and some lead introduced through the front and melted down. When this has become a cherry-red, the blast is let on and cupellation started. No distinction is made between dross, skimmings, and litharge, as in the German cupellation. The litharge is made to run off at the front, and fresh lead is supplied from the back, where one or two small bars protruding through openings into the furnace are melted down at such a rate as to keep the lead in the test always on the same level. The litharge is collected in a cast-iron pot (say 13 in. in diameter and 8 in. deep) running on wheels. With the large furnaces in common use to-day an ordinary slag-pot with a loose central partition wall has replaced the small litharge-pots. The object of the partition is to facilitate the breaking of the cone of litharge. The litharge-pot was at one time replaced by a water-box. This has two advantages. It reduces the temperature for the cupeller and presents the litharge in a granulated form, which is easily handled and sampled. It has, however, the disadvantage that a cupel-carriage cannot be used, and that in the blast furnace there is more loss in lead and silver when granular litharge is charged than if it is in lump form. Granulating has been abandoned.

With a stationary iron test ring, the litharge is run off through a gutter cut into the filling. It is not often that one gutter serves for the passage of the litharge; generally there are three and often four, opened one after the other to prevent excessive corrosion. A movable test gives an additional mode of regulating the flow of litharge by lowering and raising the front. With the

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Steitz water-jacket test the flow of the litharge is regulated only by the quantity of the lead that is melted off from the bars at the back of the furnace. The gutter can be closed for a short time by allowing litharge to accumulate there, or with a piece of clay.

The flow of the litharge is so regulated that about one-half the surface of the lead remains covered. The former practice of cupelling and fining in the same furnace has been abandoned at many works. At present it is common to concentrate the bullion to 60 or 70% of silver on one test and to fine it in a separate furnace. For this concentrating, the water-jacket test is excellent, as it can be run by an inexperienced man, while judgment and practice are necessary with a test where the litharge gutter has to be regulated by the cupeller. By thus dividing the cupelling into the two operations of concentrating and finishing, a smaller number of experienced and reliable cupellers is necessary. When the bullion is concentrated to the desired degree, it is ladled or poured out and goes to the finishing furnace, and the concentration furnace is again filled. Thus a concentrating furnace runs constantly. After a certain time the bottom becomes too thin and has to be exchanged. A test ring filled with limestone clay, if used four or five hours daily for finishing, lasts only 30 days; a cement-test used for the same purpose lasts months. A water-jacket test filled with limestone clay, used for concentrating, lasts only 60 days. A test ring filled with cement and used for concentrating and refining lasts seven days.

The finishing is always done on a test having an iron test ring. The operation is the same as in concentrating, but usually not continuous. Toward the end, when the silver-lead alloy becomes less readily fusible, the temperature has to be considerably raised. When the silver has become sufficiently concentrated the addition of rich bullion is stopped. The last litharges are drawn off and the test remains almost filled with crude silver, which has now to be fined. It is not often that the brightening is seen. Samples taken from the metal bath show how far the cupellation has progressed. The fining consists usually in exposing the silver for some time to the action of the heat and the blast. Bone-ash is sometimes given in small quantities to absorb the impurities that float on the surface or have collected on the edge. At some works nitrate of soda is used in the furnace to make the silver at least 997 fine, as this has become the standard below

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which fine silver should not go. The niter is spread on the silver, a shovelful (about 12 lb.) at a time. To prevent the soda from corroding the filling of the test ring, some refiners spread finely ground brick over the nitre. The slag, floating on the silver, is removed only when the silver is fine and ready to be cast into molds. About 15 shovelfuls of nitre are required for 50,000 oz. of silver. In this connection the behavior of tellurium is of interest. Ulke* states that in the second refining of doré silver (obtained in working the anode mud from an electrolytic copper refining plant) with nitre, the resulting slag contained as much as 20% tellurium, in the form of tellurite of sodium (see § 139).

The indications of fine silver are: a smooth, clean surface; that stirring fails to bring impurities to the surface; that a tool held over the silver is clearly reflected in it; that a sample taken by dipping in a rod will show no spots whatever on the surface, and have a pure, silver-white color; and that a sample taken with a spoon will spurt while cooling, although this is not a good test. Some refiners cast a small sample-bar, examine the surface, which should be smooth; the fracture, which should be finely granular and show a silky luster; test for malleability by hammering, etc. The only way to know definitely how the fining is progressing is to make an assay. This is done in the dry way, by weighing out twice ½ gram of a granulated sample and ½ gram of c. p. silver as a check, and cupelling the three samples with the same amount of lead on three cupels placed in a row in the muffle. A second assay half an hour later will show whether any progress has been made. An assay in the wet way, with potassium-sulphocyanide, using ferric sulphate as an indicator, will give the same result quicker than cupelling.

When the silver is fine, that is, when it ranges between 997 and 999½, it is either ladled out into warmed molds, or, if the Lynch test-support is used, it can be poured. Sometimes this is done into water, to be remelted at a lower temperature in a plumbago crucible, a new retort (heated in a tilting furnace), or a separate cupelling furnace. If in the last, the granulated silver is covered with charcoal. If the silver is to be poured or ladled from the test in which it has been refined, it is advisable, if smooth bars are to be obtained, to cover it about 20 minutes

* *Engineering and Mining Journal*, Nov. 28, 1896.

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before pouring with charcoal. While filling the molds, which have been warmed and smoked, a small stick is held in contact with the surface of the silver to collect floating slag or impurities. When a mold is filled, a small piece of paraffine is thrown on the metal and the lid quickly put on.

3. *Sampling Fine Silver.*—The sample of the fine silver is best taken from the mold. When this has been filled, a long-handled iron spoon is inserted, the silver stirred with it, and the sample taken out and poured on a wet board resting in an inclined position on the rim of a basin filled with water (Eurich). The granules collecting in the basin are very small and make weighing of a definite quantity for the assay a quick and easy operation. Taking chips from different parts of the bar is unsatisfactory, as the impurities are as a rule not evenly distributed.*

4. *By-Products.*—The by-products of the cupellation process are litharge, cupel-bottom, and fluedust. Litharge coming from retort bullion runs, when it is pure, from 50 to 60 oz. to the ton; when it is impure, *e.g.*, when drosses of the retort bullion are being scorified, often from 150 to 200 oz. The cupel bottom varies too much in lead and silver to give any average figure. Samples of fume and dust from cupelling rich retort bullion gave the following values:

	Pb. Per cent.	Ag, Oz.	Au, Oz.	Authority.
Dust	21.6	20.0	0.16	F. B. F. Rhodes.
Dust	27.6	106.6	0.40	M. W. Iles.
Gray fume	41.8	134.3	0.33	M. W. Iles.
Red fume on cupel door	59.0	1646.0	.....	M. W. Iles.

These by-products always go to the ore blast furnace; litharge is sometimes used, as already stated, to hasten the softening of base bullion that is especially hard.

5. Results.—In a test 4 ft. 6 in. by 3 ft. 6 in., 7,000 lb. of retort bullion are cupelled by two men in 24 hours, using from 1½ to 2 tons of coal, according to the quality of the fuel. On a larger test, 7 ft. by 4 ft. 10 in. and 4 in. deep, holding from 1,500 to 2,000 lb. retort bullion, 6 tons are cupelled in 24 hours with 1½ tons of coal. In a similar test, 7 ft. by 5 ft. 4 in., 6½ tons 2,000 oz. retort bullion are cupelled in 24 hours, 39 gal. of

* Blake, *Journal of Chemistry*, 1888, p. 71.

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reduced oil atomized by air being consumed for every 1,000 oz. silver produced. It is advisable to have 8-hour shifts for cupellers to prevent their becoming leaded.

The concentrating of 1,000 lb. of 70% bullion on a 33 by 28-in. test, 5 in. deep, and refining of the resulting silver (say 12,000 oz. silver), lasts about five hours, requires one man, and about 1,500 lb. of nut coal. When the finishing-furnace is stopped for a day or two, the fire on the grate is kept going in order that the temperature of the test may not sink below a dull red heat; charcoal is often kept aglow on the test, as it makes it again porous when it is much soaked with litharge.

The loss of lead in cupelling is generally given as 5%

§ 151. Comparison of Methods.—A comparison between the two methods of cupelling leads to the conclusion that the German method is, for purposes for which cupelling is generally used to-day, by far the more expensive. Although it forms litharge more rapidly, because the hearth is so much larger, it produces only a comparatively small amount of silver as the product of one operation. To remove the silver, the furnace has to be cooled, and the hearth torn out and replaced by a new one. This takes time, thus neutralizing the advantage of the quick formation of litharge, and costs much labor, fuel, and material, including a large amount of hearth material, which has to be smelted in the blast furnace for every cupellation. In the English cupelling furnace, especially with its American modifications, a cupel bottom can be made to last for weeks; the process is therefore less interrupted, and thus much expense for labor, fuel, and material saved, but it has the drawback that the litharge is always more apt to be rich and impure.

A German cupelling furnace may be in place when the resulting litharge is to be sold as such, and is therefore required to be pure, and poor in silver. The English furnace with American improvements is decidedly preferable if the bullion to be cupelled is so rich that the resulting litharge would in any case run too high in silver to be sold as such. In this case, and it is the common one to-day, it is not of much consequence whether the litharge be a little poorer or richer in silver, or if it be somewhat contaminated with impurities, as long as the advantages more than make up for such deficiencies.

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